MINING AND MILLING METHODS AND COSTS AT THE INDIAN CREEK MINE, ST. JOSEPH LEAD CO., WASHINGTON COUNTY, MO.

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MINING AND MILLING METHODS AND COSTS AT THE INDIAN CREEK MINE, ST. JOSEPH LEAD CO., WASHINGTON COUNTY, MO.[1]

by

Carl R. Christiansen,[2] Willis A. Calhoun,[3] and Walter F. Brown[2]

INTRODUCTION

This paper, which describes methods and costs of mining and milling lead ore at the Indian Creek mine of St. Joseph Lead Co., Washington County, Mo., is one of a series being prepared by the Bureau of Mines on methods and costs in various mining districts of the United States.

The Indian Creek mine is the first important producer of lead ore at depth in Washington County. The nearest similar ore deposit is in the northern part of the Southeast Missouri Lead Belt, about 25 miles to the southeast in western St. Francois County. Development of the Indian Creek deposit was begun in 1950, and by mid-1954 the property, equipped with one of the country's most modern mining and milling plants, was in full operation.

Mining is done entirely with trackless equipment, powered by diesel engines or electric motors. The hoisting shaft was the first circular, concrete-lined shaft to be sunk in the lead district. The shaft was sunk by the company without a lost-time accident.

ACKNOWLEDGMENTS

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HISTORY (2, 8, 12)[4]

Early settlers in Illinois learned from the Indians that lead was to be found in the region west of the Mississippi River. Extravagant accounts of mineral wealth reached France, which was then in possession of the territory.

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4/ Underlined numbers in parentheses refer to citations in the bibliography at the end of this report.

The French Company of the West was organized to mine large mineral grants obtained from the French Government. An expedition under Renault left France in 1719 with 200 miners and laborers and all the equipment needed for mining operations. After stopping at Santo Domingo to purchase slaves for use in the mines, the expedition proceeded to Illinois. Continuing westward, Renault's exploratory parties discovered and operated lead mines in various parts of what are now Washington, Madison, and St. Francois Counties, Mo. Mine-à-Renault was discovered in 1724 on Fourche-à-Renault Creek, near the present Indian Creek mine. Old Mines, north of Potosi, was opened about the same time. The mines were operated intermittently. Activity in the district did not reach its peak until mining was begun at Mine-à-Breton, the present site of Potosi. Francis Azor (or Breton) discovered lead ore here in 1763 while chasing a bear.

Moses Austin, who had operated lead mines in Wythe County, Va., came to Mine-à-Breton in 1797 after receiving a tract of land from the Spanish Government, which controlled the area at that time. In consideration for the grant, Austin agreed to install facilities for mining and smelting lead and to build a shot tower. All earlier mines had been shallow diggings in residual clays, but Austin sank the first shaft in solid rock to a depth of 80 feet. He supposedly produced the first sheet lead by pouring the molten metal on a flat rock. His operations supplied the Spanish arsenals at New Orleans and Havana with shot. The lead was transported to Ste. Genevieve on pack animals or in wooden carts, then shipped down the Mississippi River. It is believed that Austin produced about 9,160,000 pounds of lead during the period 1798-1816.

Before 1869, virtually the entire lead output of Southeast Missouri came from shallow workings, mainly above the water table. The ore occurred chiefly in masses or as crystal aggregates of galena in crevices and caverns in limestone. Mining of rich, disseminated lead deposits occurring at depth in St. Francois County was begun in 1864, and within a few years deposits of this type became the principal source of production.

Up to 1869 the output from shallow deposits in Washington County was 124,930 tons of lead, valued at $5,082,700 (2). No further lead mining of consequence was done in the county until the Indian Creek operation near Potosi was started. The region about Potosi has for years been noted for its barite deposits and is one of the most important sources of barite in the United States.

LOCATION AND PHYSICAL FEATURES (2, 4)

The Indian Creek mine is in the SE1/4 sec. 3, T. 38 N., R. 1 E., Washington County, Mo., 11 miles northwest of Potosi, the county seat. (See fig. 1.) The mine plant and mill are on State Highway 114, which connects Potosi and Sullivan. The nearest railroad, at Potosi, is the terminus of a spur from a main line of the Missouri Pacific RR. at Mineral Point.

The topography of the region, which is in the stage of early maturity, is characterized by steep hills, narrow divides, and V-shaped valleys. The maximum relief is about 500 feet. The main shaft site is at an altitude of 1,058 feet, near the crest of a drainage divide.

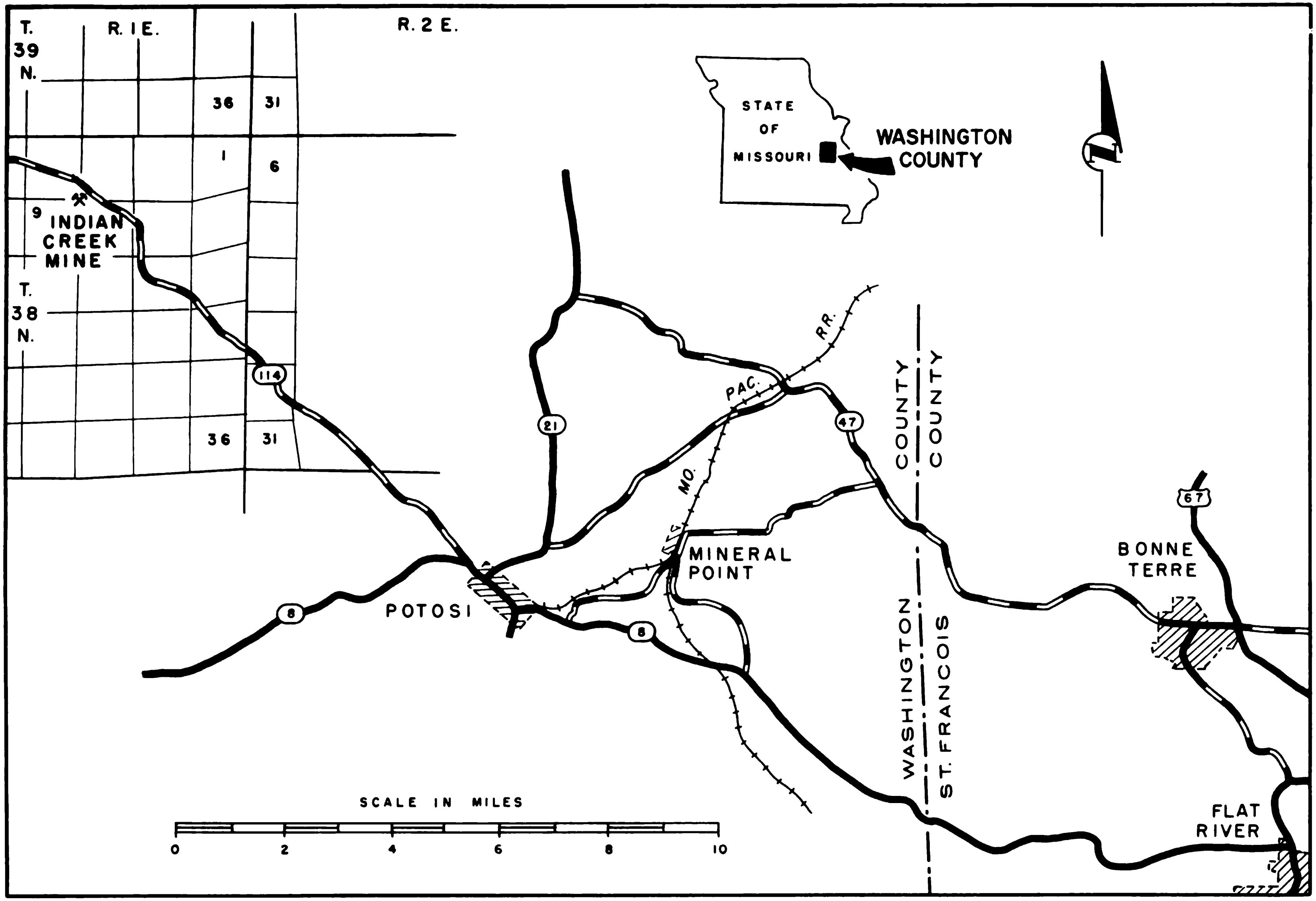

FIGURE 1. - Location Map, Indian Creek Mine, Washington County, Mo.

Potosi, with a population of about 2,400, is the residential and business center for most of the mine employees. Because the Indian Creek mine is far from the nearest St. Joseph Lead Belt operations, it was necessary to employ and train local residents for work in the mine and mill and to establish the operation as a separate division of the company, complete with shops, offices, and warehouses.

DESCRIPTION OF DEPOSITS (7, 9, 10, 11)

The ore at the Indian Creek mine is galena associated with small quantities of sphalerite and marcasite disseminated in Bonneterre dolomite of Upper Cambrian age.

The main controlling feature of the lead-zinc mineralization was a domal or anticlinal structure related to a buried Precambrian rhyolite-porphyry ridge. The eroded surface of the porphyry was buried progressively by Upper Cambrian sediments, beginning with the Lamotte sandstone. At Indian Creek, as in other Southeast Missouri areas, the higher Precambrian topographic features were not completely covered by the Lamotte formation, which wedges out against the ridges and, in turn, is covered by the Bonneterre dolomite. The line along which the Lamotte wedges out against the porphyry is called the pinchout, and ore deposition occurred in the Bonneterre formation above this line. Thus the mineralized area is in the form of an irregular band, roughly paralleling the crest of the buried ridge. (See fig. 2.)

Where ore bearing, the Bonneterre formation comprises two zones. The main body of the formation is dense, massive dolomite. Near its central part the formation consists of an irregularly bedded, mottled, fingered dolomite, believed to be of algal origin. Both zones are relatively free from vugs, vaults, joints, and other structural weaknesses. Table 1 shows the geologic column at the main shaft.

The ore deposit developed at Indian Creek is more than 4,000 feet long and has a maximum width of 600 feet. Drilling indicates that the ore-bearing zone is as much as 150 feet thick in places; however, where the zone is uncommonly thick the ore occurs in layers separated by waste. The total thickness of ore in the deposit ranges from 8 to about 60 feet. Individual ore bodies are irregular in size, shape, and dip, necessitating selective mining to avoid inclusion of excessive waste rock. Development on two or more levels is required to mine the separate ore horizons.

METHODS OF PROSPECTING AND EXPLORATION (9)

Prospecting for lead ore in Southeast Missouri, which is done almost entirely by drilling, is based largely upon a search for sedimentary structures favorable to ore deposition, termed "structural centers." These arched depositional highs are common in the Bonneterre formation and generally trend northeast-southwest. The bulk of the ore mined in Southeast Missouri occurs in the rock forming the limbs or crests of such structures. In some places, as at Indian Creek, the close relationship between the pinchout of the Lamotte sandstone and the ore zones serves as a guide to the location of ore bodies.

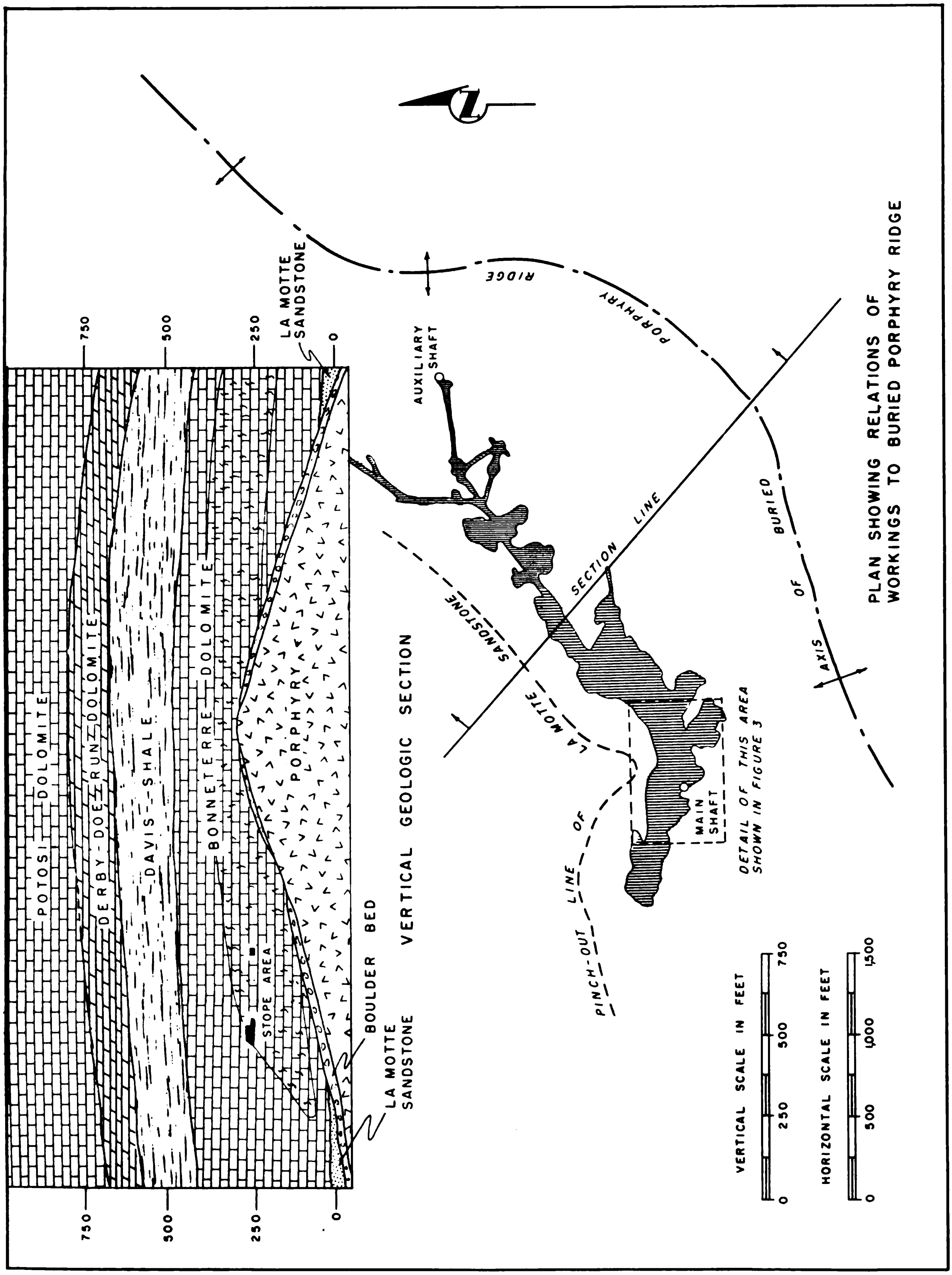

FIGURE 2. - Plan and Section, Showing Relationship Between Indian Creek Ore Body and Geologic Formations.

TABLE 1. - Geologic column at main shaft

Age	Formation	Altitude	Depth, feet[1]		Description
			From-	To-	
Recent	Soil	1,058	0	60	Clay and chert alluvium.
Cambrian	Potosi	998	60	345	Dolomite, partly oxidized and containing some moderately heavy quartz druse; some large vugs.
Do.	Potosi-Derby	713	345	442	Fine-grained, well-stratified dolomite; no quartz druse.
Do.	Derby-Doe Run	616	442	485	Dense, thin-bedded, fine-grained, argillaceous dolomite.
Do.	Davis	573	485	655	Alternating beds of shale, silty dolomite, limestone, and dolomite sandstone.
Do.	Bonneterre	403	655	762	Finely porous dolomite; mostly massive and irregular bedding, with few thin-bedded sections.
Do.	do.	296	762	884	Irregularly bedded, fingered dolomite.
Do.	do.	174	884	916	Dense, marblelike dolomite with few porphyry pebbles.
Do.	do.	142	916	931	Residual boulder bed.
Precambrian porphyry		127	931	948	Solid rhyolite porphyry.
		110	[2]948		

[1] Water table at depth of 70 feet.
[2] Bottom of shaft.

The Indian Creek deposit was discovered by prospect diamond drilling. More than 400,000 feet of prospect holes was drilled over a period of 4 years in Washington County before the first "payhole" was drilled in April 1948. By April 1950 enough ore had been disclosed to warrant development of the deposit.

The term "prospect hole" is used to describe all holes drilled to gain information from virgin ground or ground adjacent to stoped areas.

The surface prospect holes drilled to outline the ore bodies in Washington County are not put down on a standard pattern but are spotted by the geologist in conformity with his judgment. The holes usually are spaced at 300- to 600-foot intervals.

Prospecting and exploration by drilling are done continuously, both on the surface and from underground workings. A diamond-drilling contractor and two contract churn-drilling crews work on vertical holes from the surface in the Indian Creek area. One diamond drill is used underground for drilling holes laterally from mine openings. Hammer drills are used occasionally to test areas above the roofs and below the floors of stopes for ore.

In drilling from the surface, churn drills are used to penetrate the Potosi formation, which cannot be core-drilled satisfactorily because of sporadic occurrences of chert. The underlying formations are relatively uniform in hardness and are cored without difficulty. Using AX-size coring tools with a 20-foot, double-tube, rigid-type core barrel, the drillers recover more than 97 percent of the core. The high core recovery makes it unnecessary to save the sludge for assay. The core is boxed by the driller, using depth markers to aid the geologist in logging the hole. Drilling is discontinued when the Precambrian porphyry or Lamotte sandstone is reached.

Holes drilled from underground workings with a diamond drill are directed at low angles above or below the horizontal. They serve to delineate known ore bodies and prospect for additional ore within certain limits. Such drilling is essential to control of the development program because of the irregularity of the mineralized zone. A pneumatic, column-mounted drill and AX coring tools are used. A hole is begun with a 2-foot core barrel and continued with a 10-foot, AX, double-tube, rigid-type core barrel. Holes may be as long as 200 feet, and core recovery usually is nearly 100 percent. One man operates the machine on contract and averages 55 feet of hole a day.

Hammer-drill prospecting is done with airleg drills or sinkers. Holes are drilled to depths of 15 feet with tungsten-carbide-tipped bits. Separate samples of the cuttings are taken for each 3-foot steel-change interval. "Upholes" are drilled wet, and the sludge falls on waxed-paper liners in dynamite boxes. "Downholes" are drilled dry; part of the cuttings fall into a metal pan that partly surrounds the drill steel when it is in position. No attempt is made to recover all cuttings from hammer-drill holes, as it is assumed that the part collected is representative. The samples are sacked and delivered to the geologist for analyses.

The geologist logs all diamond-drill core and saves a representative section of each type of rock penetrated. These sections, labeled to indicate the depth interval represented, are stored permanently in steel drawers in a heated coreroom. Half of the core for each mineralized section is saved and the other half assayed for lead, zinc, and copper. The location of each prospect drill hole and all pertinent data are shown on a plan map.

The geologist visually analyzes the cuttings from hammer-drill holes by panning them. If the estimated lead content is over 1 percent, the sample is assayed.

Table 2 shows statistical data on prospect drilling at Indian Creek since mining was begun.

TABLE 2. - Prospect drilling

				Underground drilling			
	Surface drilling			Core		Hammer	
Year	Number of holes	Core	Churn	Number of holes	Feet	Number of holes	Feet
1954	-	-	-	87	11,277	179	1,284
1955	-	-	-	56	9,101	140	1,338
1956	24	11,234	8,756	44	10,024	157	1,494
1957[1]	21	9,871	3,790	39	7,395	61	667

[1] To July 1.

METHODS OF SAMPLING AND ESTIMATING TONNAGES AND VALUES

Ores are classed as commercial and low grade. Commercial-grade ore contains a minimum of 2 percent lead, 4 percent zinc, or lead and zinc of equivalent value. Ore having a metallic content lower than the cutoff for commercial ore but containing not less than 1 percent lead, 2 percent zinc, or both lead and zinc of an equivalent value is considered low grade.

Core-drill holes are designated, according to assay values, as follows:

(1) Pay hole: One having a minimum average grade of 2-1/2 percent lead and a minimum content of 20 feet-percent lead within an interval of 8 feet.

(2) Indicative hole: One having an assay between 1 and 2-1/2 percent lead and a minimum content of 8 feet-percent lead within an interval of 8 feet.

(3) Trace hole: A mineralized hole lower in grade than (2).

(4) Blank hole: A hole with no indication of mineral value.

The information furnished by the irregularly spaced exploratory drill holes at Indian Creek served as the basis for estimating ore reserves of the deposit. Reserves were computed by the polygonal-prism method for extensive mineralized areas containing ore of commercial grade. In smaller areas blocks of ore were resolved for individual holes or for groups of holes having correlative ore strata. Data from holes drilled at the surface are supplemented by information from underground development and drilling.

As of July 1, 1957, the estimated reserve of unmined ore in the Indian Creek deposit was approximately 4,300,000 short tons with a weighted average grade of 3.82 percent lead and 0.35 percent zinc. Past experience indicates that a greater tonnage of ore of lower grade will be mined from the deposit.

No face or bulk samples are taken from underground workings to determine the grade of ore. The shift foreman and mine captain make visual estimates of the percentages of lead in the working faces and thus determine which faces should be mined. Each week an estimate of the tonnage and grade of ore produced in each stope and development opening is sent to the engineer's office; the tonnage is approximated by multiplying the truck capacity by the number of loads. Hoisted ore is weighed on reaching the mill, then sampled for assay. The difference between the actual and estimated total tonnages is prorated to the various working places, and corrections are made in the estimates of metal content of the ore.

METHODS OF DEVELOPMENT AND MINING

Ore is mined in open stopes with irregularly spaced pillar support. A 3-compartment, circular hoisting shaft and a circular auxiliary shaft, 3,000 feet apart, provide access to the mine. The hoisting shaft was sunk where preliminary drilling indicated it would reach the lowest part of the ore deposit. Later development and exploration disclosed ore at lower levels; however, the bulk of the ore can be hauled downgrade to the shaft station, and most of the water in the mine drains by gravity to the main sump near the hoisting shaft.

The 2 shafts are connected by a haulage drift, which has an ascending grade of 2 percent from the hoisting shaft. Other drifts radiate from the shaft station or branch off from the main haulageways to the various stopes. In parts of the mine where ore horizons are separated by lean or barren rock, the ore is developed and mined on two or more levels. (See fig. 3.)

Most of the broken ore is loaded into trucks at the face and hauled directly to the shaft station, but in some places transfer raises are driven to upper levels to reduce haulage distances or avoid hauling over steep grades. The ore from an upper level southeast of and not far from the hoisting shaft is passed through an inclined raise to the main level and scraped from there to the ore pocket. In the northeastern part of the mine an upper level was developed from an incline driven on a 12-percent grade from the main level. A drift has been driven beneath the upper level and connected with it by a raise, which is to be used as an ore pass.

As the ore-waste contact in the floor of the stopes is irregular, it is often necessary to leave some ore in the floor or to mine some waste to obtain the desired road grade for ore haulage. This ore is recovered later.

Development

Shaft Sinking (1, 6)

The main and auxiliary shafts are circular, and each has an inside diameter of 12 feet 7 inches and a concrete lining 15 inches thick. The lining and steel guides for the work cage were placed as shaft sinking progressed. Pipes to be used as conduits for air, water, and electricity were enclosed in the concrete lining. Sections of the completed main shaft are shown in figure 4.

The permanent headframe of the main shaft was erected before sinking was begun. A concrete collar 10 feet deep and 24 feet in diameter was poured August 7, 1951. A sliding steel door, riding on rails and operated by a compressed-air cylinder, was installed at the collar. It was used to cover the shaft and thus protect the shaftmen while rock was being dumped.

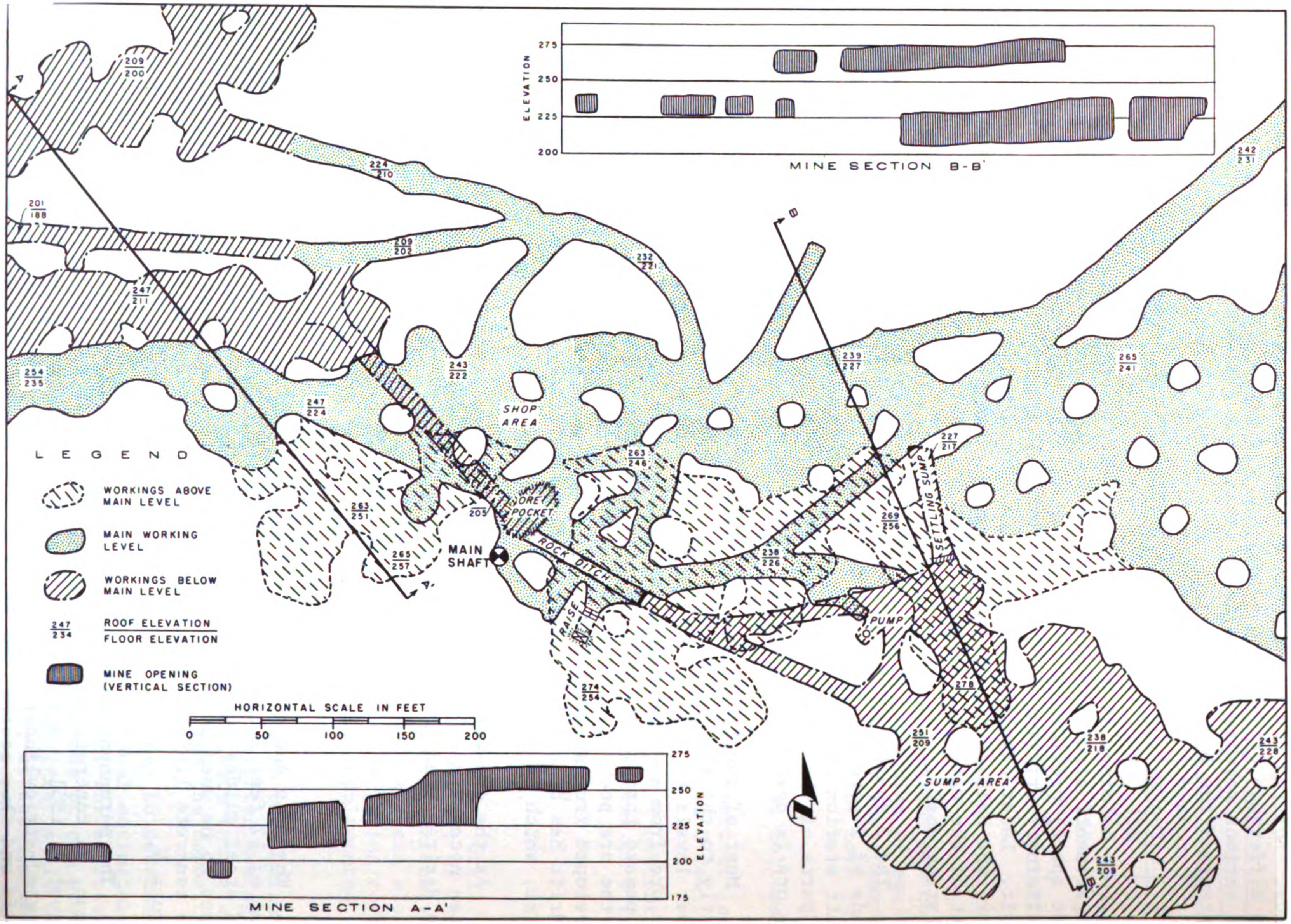

FIGURE 3. - Plan and Sections of Mine Workings Near Main Shaft.

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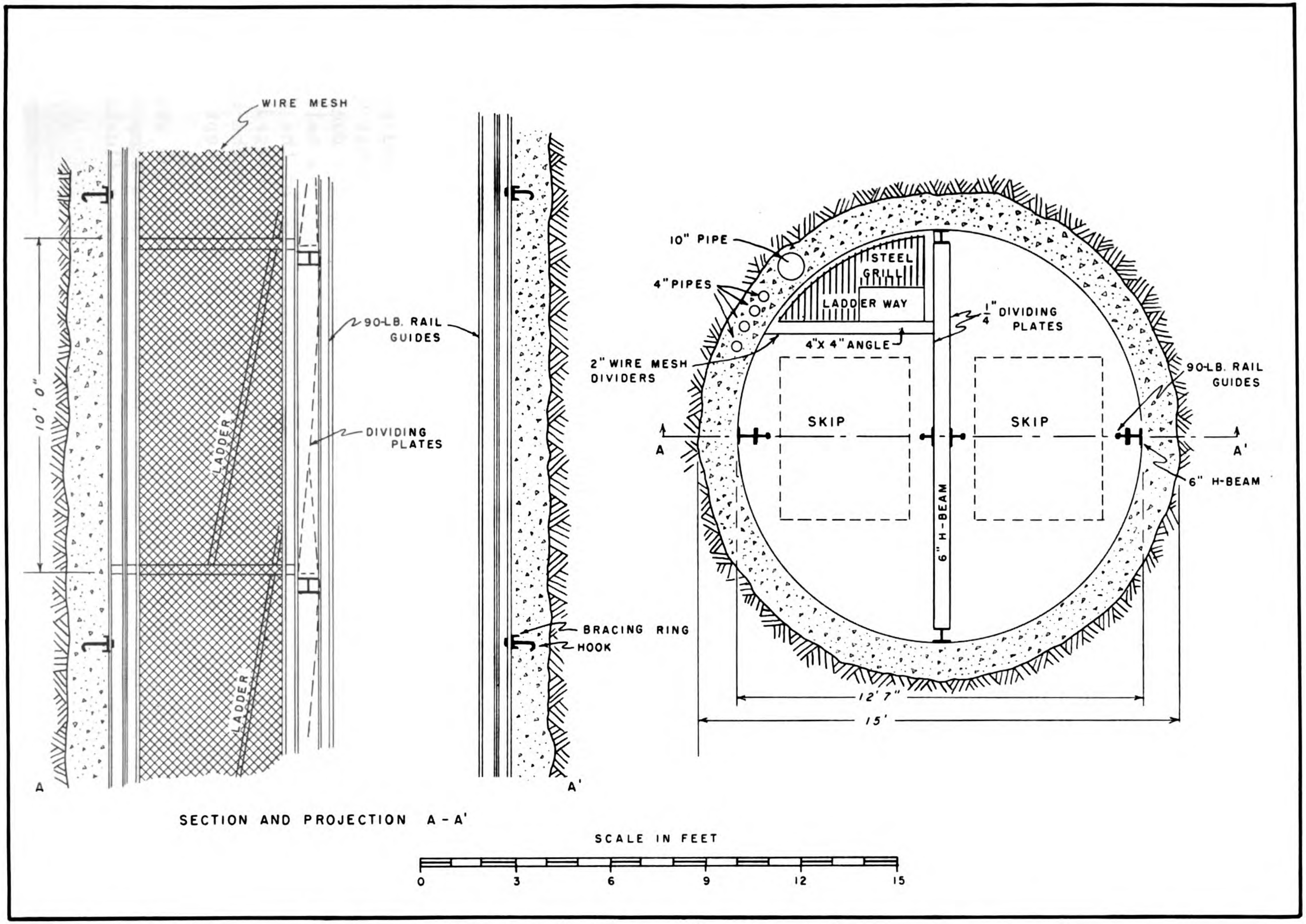

FIGURE 4. - Sections of Hoisting Shaft.

A 7-man crew worked on each of the three 8-hour shifts. A crew comprised a shaft foreman, hoistman, maintenance man, and 4 shaftmen. The shaftmen did the drilling, blasting, mucking, setting of forms, and concreting; each man was familiar with all phases of the work.

A drill round in the shaft consisted of 56 holes drilled in 4 concentric rings. (See fig. 5.) Drilling was done with a 4-machine jumbo designed especially for the job. Joy LM-67 drifters with 7-foot feeds were mounted on arms extending from the center of the jumbo. A fixed angle was maintained between the feed and the arm for all holes to assure that each hole in a ring would be at the proper angle from the vertical. The correct azimuth of each hole in the round was obtained by an indexing device consisting of holes punched in the jumbo bonnet and alined with a pointer on the movable drill arm. The drill arms were raised or lowered to drill another ring of holes, while the fixed angle was maintained between the feed and the arm. Each machine drilled all the holes in its assigned quadrant. Most rounds were drilled to a depth of 8 feet; however, by changing the angle between feed and arm, the crew could drill rounds of other depths.

The eight holes of the innermost ring of the shaft round were drilled to meet at the bottom, making a conical cut. The cut holes were surrounded successively by two rings of relievers and a trimming ring. All holes were 1-5/8 inches in diameter and were drilled with carbide-insert detachable bits on 6- and 10-foot steels. They were charged with semigelatin dynamite having a weight strength of 65 percent. The rounds were fired electrically, using millisecond-delay caps. The best fragmentation was obtained by alternating instantaneous and No. 1 delay caps in the cut, Nos. 3 and 4 delay caps in the first ring of relievers, and Nos. 6 and 7 delay caps in the second ring of relievers and by using No. 9 delay caps on the trimmers.

The shaft was mucked mechanically with a 3/8-cubic-yard clamshell operated by two air-motor-powered hoists permanently installed on the surface near the shaft collar. The air hoists were equipped with spring-loaded brakes and solenoid controls, thus permitting a shaftman to stand on the muckpile and operate them by remote control. Two ropes attached to opposite sides of the clamshell permitted the workmen to guide it laterally for digging and dumping. (See fig. 6.) The rock was mucked into a 54-cubic-foot bucket. The hoisted rock was discharged onto a dumping door hinged to the bottom of a slide that was attached to the headframe. The door was lowered into position for dumping after the bucket passed above it. As the loaded bucket was lowered, a ball and chain attached to its bottom engaged in a slotted projection of the door; further lowering caused the bucket to dump its load. The discharged rock fell into a chute and was removed with a double-drum scraper. While the bucket was above the shaft collar, the sliding, horizontal, shaft door was kept closed.

Breakage by a shaft round formed a sump in the center of the shaft bottom. This made cleanup easier, as the clamshell worked well in this sump and small pieces from around the walls could be readily scraped or water-flushed into the sump.

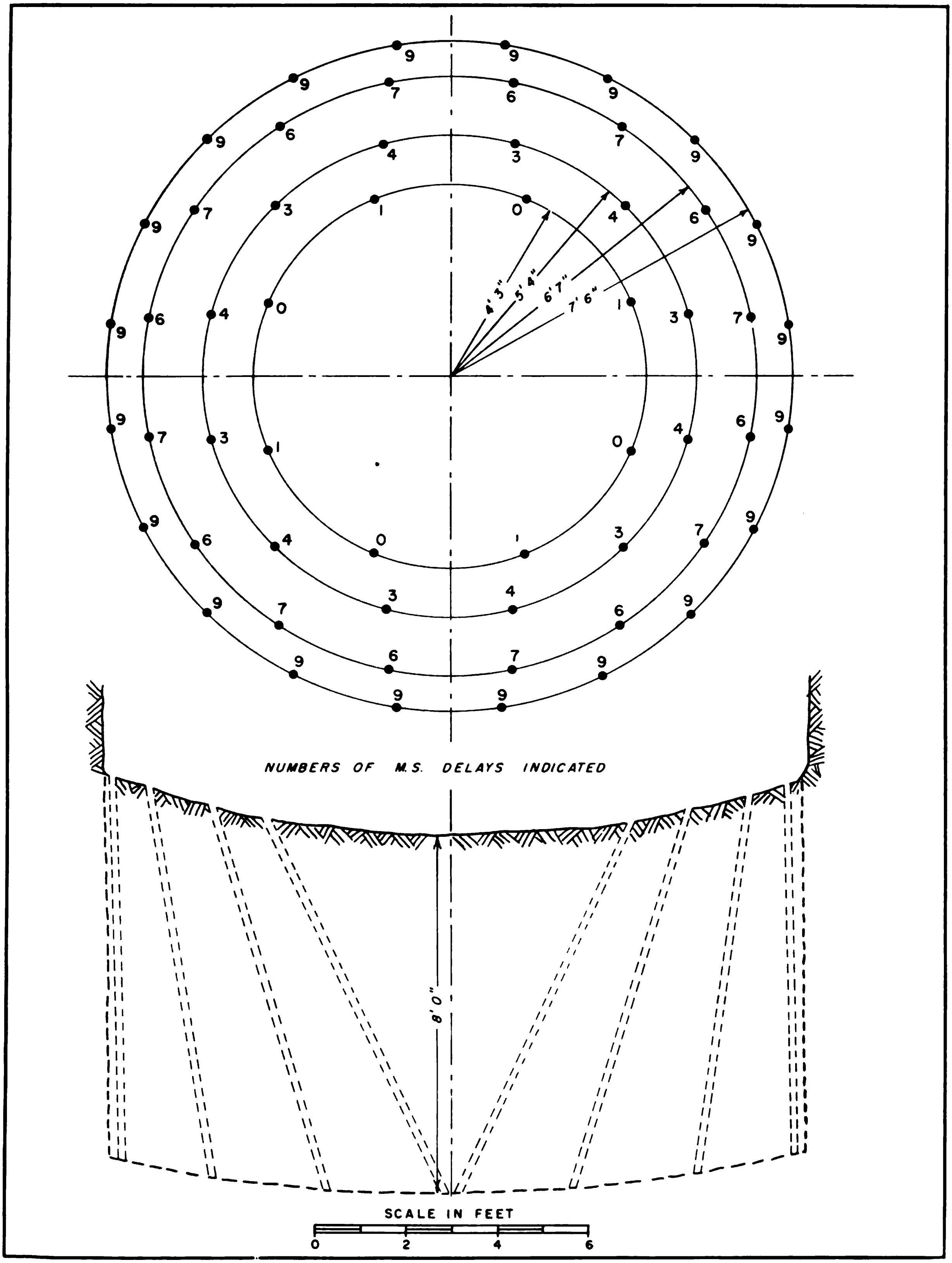

FIGURE 5. - 8-Foot Shaft Round.

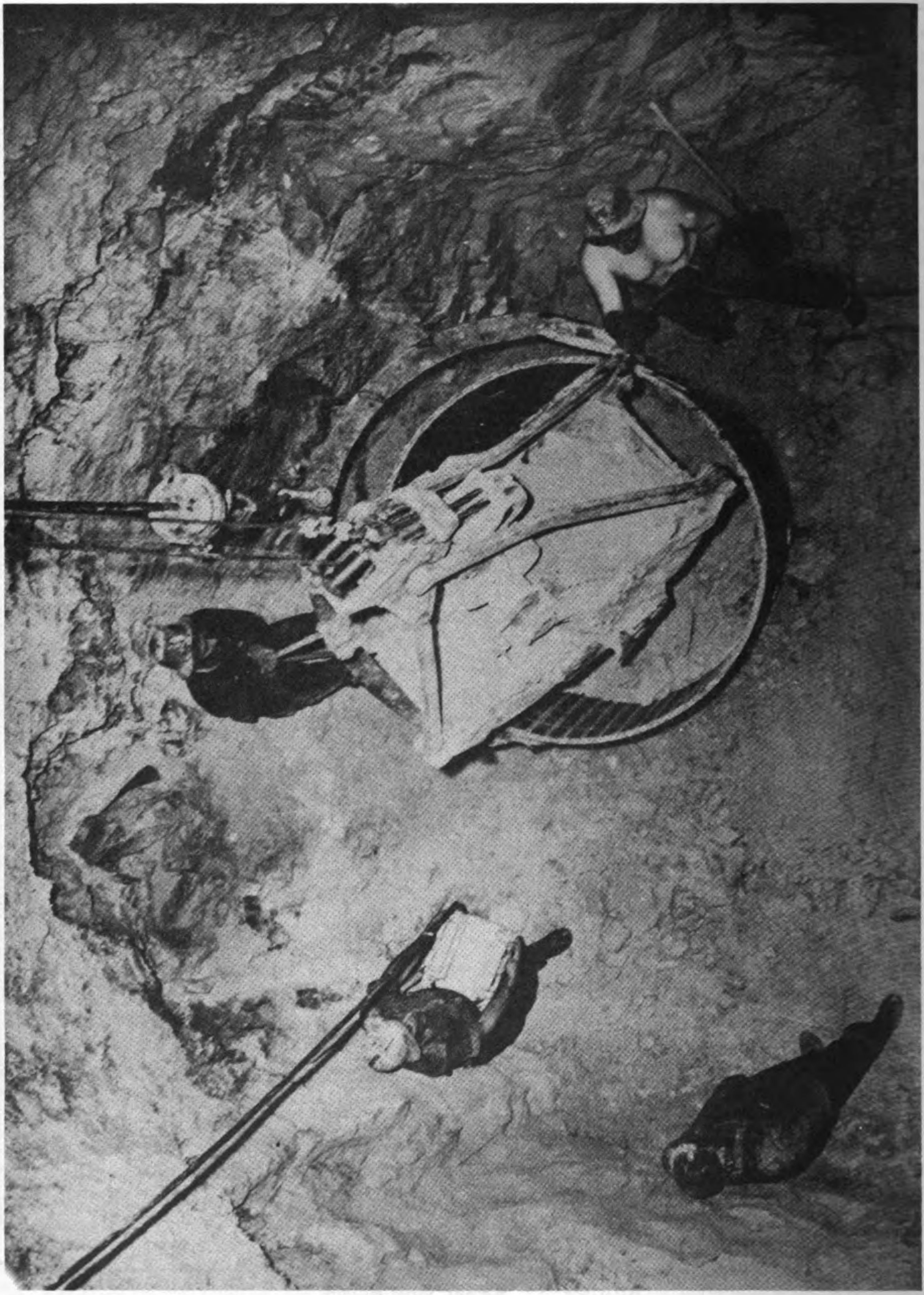

FIGURE 6. - Mechanical Shaft Mucking.

The concrete lining was poured in 20-foot sections, usually after 3 drill-blast cycles were completed; however, shorter sections could be poured. When the shaft bottom was about 30 feet below the concrete, the broken rock from the last blast was leveled as a base for the forms and a template for the accurate alinement of four 6-inch, wide-flange beams. When installed, the beams were equally spaced in the circumference of the shaft and anchored to the concrete lining by hook bolts. They served as guides for a work cage during sinking operations and afterward as supports for skip guides. With the aid of the template, beam sections were accurately plumbed and jacked solid. Lengths were added to one 10-inch and four 4-inch pipes for enclosure in the concrete. A circular steel form in 4 quadrant sections, suspended on 5/8-inch cables from winches at the shaft collar, was lowered and jacked into place; the steel beams afforded added stability to the form. (See fig. 7.) Concrete was prepared with a 1-1/4-cubic-yard mixer and lowered in a bottom-dump bucket of 1-1/2-cubic-yard capacity for pouring.

After the concrete had set, the jacks and other form supports were removed, permitting free access of drilling and mucking equipment. The forms were left in place until time for the next pour to protect the green concrete against blasting.

The work cage had a double deck and was made of steel. It was designed especially for use in placing the forms and providing internal bracing for them during concreting.

Ventilation was provided by connecting a positive-displacement blower, furnishing 4,000 cubic feet of air a minute, to the surface end of the 10-inch pipe in the shaft lining.

Floodlights were used to illuminate the shaft bottom, eliminating the need for cap lamps except when holes were loaded. All electrical control wires were suspended from the steel cable and canopy, which could be raised as desired.

Before shaft sinking was begun, two churn-drill holes were put down alongside the shaft site and equipped with deep-well pumps to control drainage. This plan did not prove to be effective, and sump pumps were required to raise the inflow water to pumps stationed at higher levels in the shaft for relaying to the surface. Later, after about half of the shaft had been completed, a diamond-drill-hole connection was made between the shaft and one of the churn-drill holes. Water was pumped from the shaft bottom through the diamond-drill hole into the churn-drill hole and raised to the surface with a deep-well pump. Pressure grouting in advance of sinking sealed off water in exceptionally wet strata.

Shaft sinking was begun November 20, 1951, and completed at a depth of 948 feet September 14, 1952. The shaft was then divided into two hoisting compartments and a ladderway compartment. (See fig. 4, p. 11.) The steel used for this purpose was prefabricated for bolting to the wide-flange beams. All steel inside the shaft, including dividers, dividing plates, ladderway, and guides, was installed in 30 shifts.

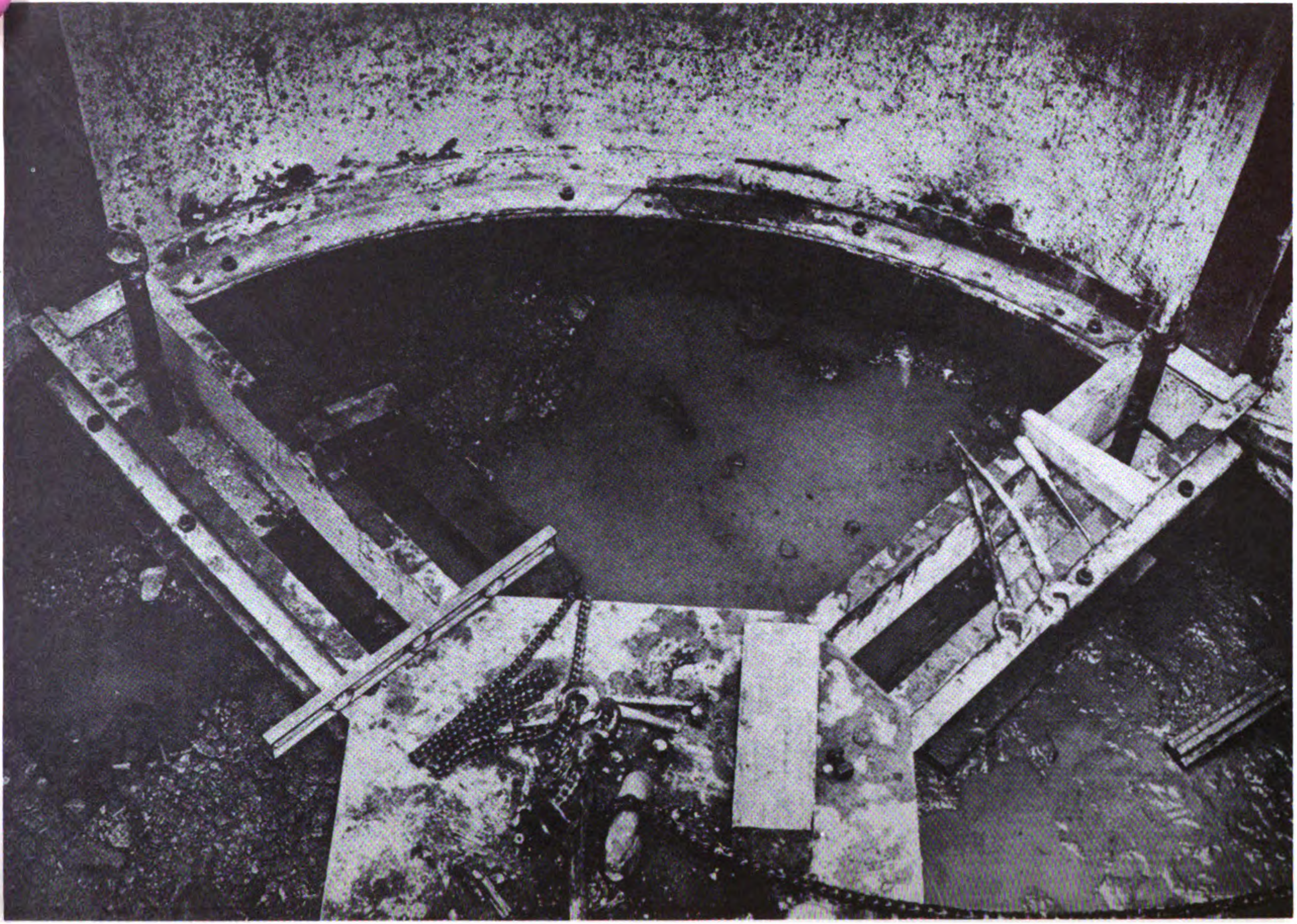

FIGURE 7. - Template and Forms for Concrete Shaft Lining.

The auxiliary shaft was sunk with the same equipment and procedures as the main shaft, except that no churn-drill holes and deep-well pumps were used for drainage. Shaft sinking was begun December 6, 1953, and completed May 1, 1954. It was sunk for $100 per foot less than the main shaft owing to the experience acquired on the earlier operation. Table 3 gives pertinent data on sinking the auxiliary shaft.

TABLE 3. - Pertinent data on sinking of auxiliary shaft

Date started: Dec. 6, 1953 Date completed: May 1, 1954

Diameter of rock excavation	feet	15
Inside diameter of shaft	do.	12.6
Total depth of shaft	do.	705
Excavated	do.	695
Concreted	do.	651
Number of rounds drilled		97
Advance per round, average	feet	7
Number of times forms were set		34
Buckets loaded:		
By hand		570
By machine mucker		4,964

Time distribution

Type of work	Shift-hours Worked	Shift-hours Delays	Total shift-hours
Drilling	346-1/2	14-1/2	361
Charging, blasting, and ventilation	212	4	216
Mucking	[1]1,262	67	1,329
Placing forms	457	32	489
Concreting	272-1/2	9	281-1/2
Grouting			356
Pump stations			191

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[1] Consists of 418-1/2 hours of hand mucking and 843-1/2 hours of machine mucking.

Lateral Development

After the main shaft had been bottomed, development work comprised cutting the ore pocket and skip-loading station; enlarging the shaft station; and driving drifts to develop the ore to the northwest, northeast, and southeast. A ditch was cut in the floor of the opening to the southeast to serve as a storage area for waste rock. This ditch was connected with the ore pocket, into which a scraper could move the rock for hoisting. (See fig. 3, p. 10.)

Construction of the ore pocket was begun by driving an inclined raise from the shaft, 50 feet above its bottom, to the level of the proposed shaft station. Later, the raise was enlarged to complete the ore pocket. A temporary chute was placed in the bottom of the raise so that development rock could be loaded directly into a bucket for hoisting. Upon completion of the raise, excavation of the shaft station and development drifts was begun. A triple-drum scraper was installed to drag the rock to the ore pocket. As the distance between the faces of the drifts and the shaft increased, drag scraping became impractical and scraper ramp loaders and 12-ton electric shuttle cars were lowered into the mine to take over the loading and hauling of development ore and waste.

Drift rounds at Indian Creek originally were drilled with drifters mounted on vertical columns, but currently development drilling is done with airleg drills or a Joy Drillmobile. Initially the drifts are driven 9 feet high and 12 feet wide, or 11 feet high and 14 feet wide. A modification of a horizontal V-cut round is used for all drift headings. (See fig. 8.) The holes are loaded with 1-inch, semigelatin dynamite of 65 percent weight strength. The main haulageways in the mine are 11 by 24 feet in cross section to permit safe truck traffic in 2 directions. An opening of this size is made by drilling slabbing holes parallel to the sides of a smaller drift to break to the desired final cross section. Three vertical rows of slabbing holes having a 3-foot burden on each row are required to enlarge an 11- by 14-foot drift.

Two development crews are employed on the day shift and 2 on the night shift; each crew consists of 2 men who perform all work necessary to advance the drift. Usually a cycle of operations is completed each shift; after the rock from the blast of the previous shift has been mucked, a new round is drilled, then blasted at the end of the shift. Each development crew uses a compressed-air-powered Eimco rocker shovel mounted on Caterpillar treads and a 6-ton truck. One crew member drives the truck and the other operates the loader.

Development crews work on contract, receiving bonuses based upon the feet of advance in excess of an established base. When slabbing is done to enlarge a drift, the work is expressed as "equivalent drift-feet of slabbing" (table 4) and the bonus paid accordingly. An equivalent drift-foot is defined as the amount of slabbing accomplished with the same effort as that required to drive 1 foot of drift. Table 4 gives the record of development at the Indian Creek mine.

TABLE 4. - Development data

Year	Drifts, feet	Raises, feet	Shafts, feet	Total development, linear feet	Equivalent drift-feet of slabbing
1951	-	-	263	263	-
1952	187	152	685	1,024	357
1953	2,696	9	131	2,836	2,754
1954	3,593	-	585	4,178	1,305
1955	1,477	50	-	1,527	2,116
1956	1,709	58	-	1,767	1,330
1957[1]	1,098	23	-	1,121	1,104
Total ..	10,760	292	[2]1,664	12,716	8,966

[1] To July 1.

[2] Includes excavation of sump in auxiliary shaft.

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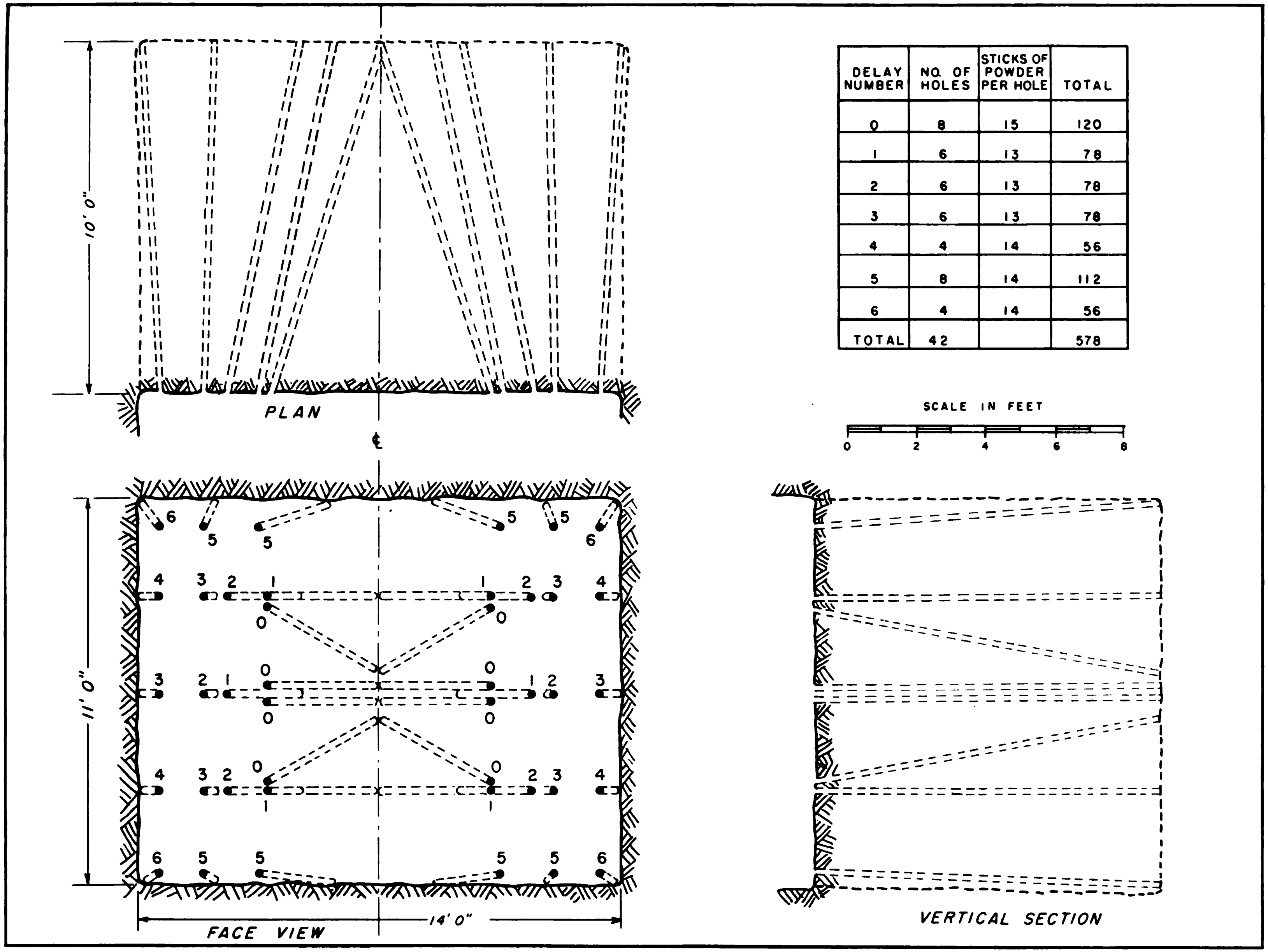

DELAY NUMBER	NO. OF HOLES	STICKS OF POWDER PER HOLE	TOTAL
0	8	15	120
1	6	13	78
2	6	13	78
3	6	13	78
4	4	14	56
5	8	14	112
6	4	14	56
TOTAL	42		578

FIGURE 8. - Standard 11- by 14-Foot Drift Round.

Mining

The mine operates on a two-shift schedule, with the same number of production men on each shift. About 2,000 tons of ore is mined a day, which is the present capacity of the mill.

Owing to the irregularities of the deposit, it was necessary to select equipment and a mining method that would permit maximum flexibility. Basically, the mining method is open stoping with pillar support. Pillars are 20 feet in diameter and spaced approximately on 55-foot centers. The ore-bearing rock is dense, strong, and almost free from structural weaknesses; hence, no support other than pillars is necessary.

Stopes range in height from 10 to 55 feet. Simple breast stoping is done where the ore is not too thick, and a breast-and-bench method is used in the thicker parts of the ore deposit.

Drilling and Blasting

All drilling in stopes is done with airleg drills equipped with 7/8- by 4-1/4-inch chucks. Hollow, 7/8-inch, hexagonal drill steel is used, with integral tungsten-carbide chisel bits. Drill steel ranges in length from 2 feet 7 inches to 15 feet 9 inches, and the respective bit gages range from 1-11/32 to 1-5/32 inches.

The ore in the Indian Creek mine is not highly abrasive. Bits can be used throughout the life of the steel without resharpening. Usually about 2,000 feet of hole can be drilled with a piece of steel.

The unit costs of drilling in 1956 were as follows:

	Cost per ton
Operating labor	$0.300
Power	.021
Maintenance and drill·steel	.043
Total	.364

All blasting in the mine is done with semigelatin dynamite of 65-percent weight strength. The cartridges are 1 inch by 8 inches. Regular delay blasting caps, fired with portable blasting machines, are used to detonate the explosive charges in face rounds. Caps and fuse are used in secondary blasting. Boulders may be broken at the beginning of the lunch period, but all other blasting is done at the end of the shift.

Explosives are stored near the stopes. The shift foreman inventories the supply each week and determines the quantity used by each miner. The miners break ground on contract and are charged for the explosives used.

The average cost of blasting, per ton of ore broken, was $0.114 in the first half of 1957. Ore breaking in the same period averaged 75.9 tons per driller per shift - an increase of 29.5 percent over the unit breaking rate of 58.6 tons in 1956. The improvement can be attributed to experience gained by the drillers, most of whom were novices when first employed at Indian Creek, and to the opening of larger stopes where greater tonnages of ore can be broken per unit of drilling.

In mining relatively thick ore by breast-and-bench stoping a breast 9 to 11 feet high is advanced at the top of the ore ahead of the benches. Slabbing rounds are used to form and maintain a saw-toothed face for advantageous drilling and breaking. (See fig. 9.) Holes are drilled nearly parallel to the free face, leaving a burden of 3 feet. A slabbing round in a section of the breast usually consists of three 12-foot holes drilled in a vertical row. The center hole is fired instantaneously and the other two with No. 1 delay caps. Two vertical rows of slabbing holes may be drilled to increase the length of a single advance.

After the breast has been advanced 10 feet or more ahead of the face of the top bench, "splitter" holes are drilled into the benches and blasted. These holes usually are 12 feet long and spaced 6 feet apart in 2 parallel, horizontal rows (fig. 9). Each row of holes carries a burden of 3 feet. The upper row is fired instantaneously and the lower row with No. 1 delay caps. If the face of the bench is badly fractured, "stope" holes may be drilled from the top of the bench to break the ore (fig. 9). A bottom bench is drilled in the same manner as a higher bench, but a third row of "lifter" holes is drilled for breaking to the desired floor level. If advantageous, the bottom bench is broken first and the next higher bench drilled from the muckpile.

Care is used in turning pillars to avoid fracturing the supporting rock. Short slabbing holes with light burden are drilled, sometimes perpendicular to the pillar but usually tangential to it, to achieve the desired breakage.

The elevation of the top of the ore usually is irregular. It is impracticable to follow major projections of ore into the back in breast stoping, so this ore is left in place while the main body of ore is being mined. The high ore is mined later by back slabbing. Starting at the breast, upholes are drilled into the ore and blasted. The broken ore is allowed to accumulate on the floor of the stope so that it can be used as a base for drilling progressively higher slabbing rounds. Slabbing holes are drilled horizontally or as upholes from the muckpile. A platform mounted on a hydraulic crane is also used for back slabbing roof ore. (See fig. 10.) The drilling platform can be elevated as high as 36 feet above the floor. One driller operates an airleg drill on the platform.

All drillers work on contract and receive bonus payments that vary according to the type of work performed. Different bonus rates have been established for breast stoping, back slabbing, taking up bottom, and other work. A separate bonus is figured for each stope.

Loading (3)

Ore broken in the stopes is loaded into trucks with portable scraper ramps. (See fig. 11.) A loader of this type was selected because of its

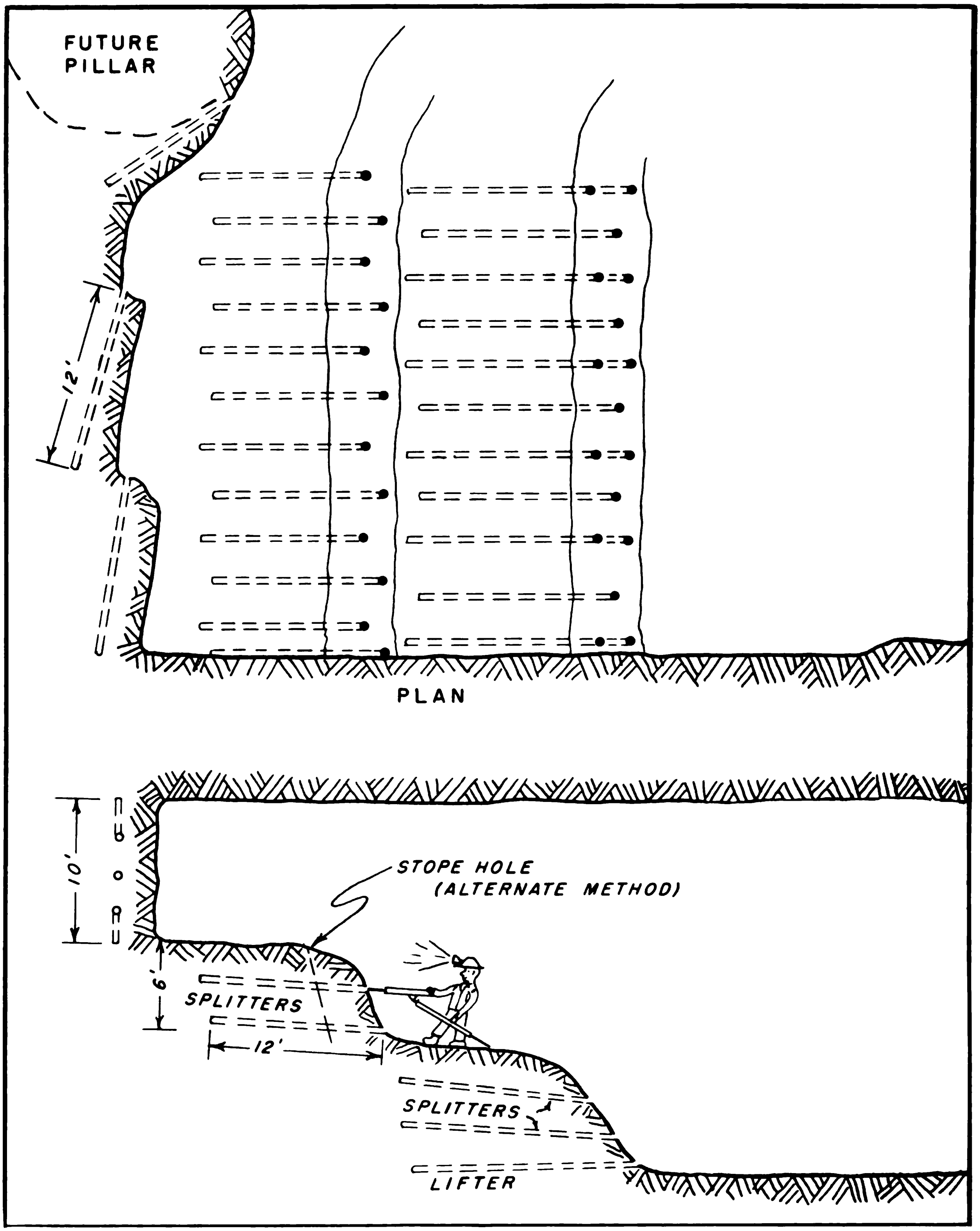

FIGURE 9. - Drilling Layout for Breast-and-Bench Stoping.

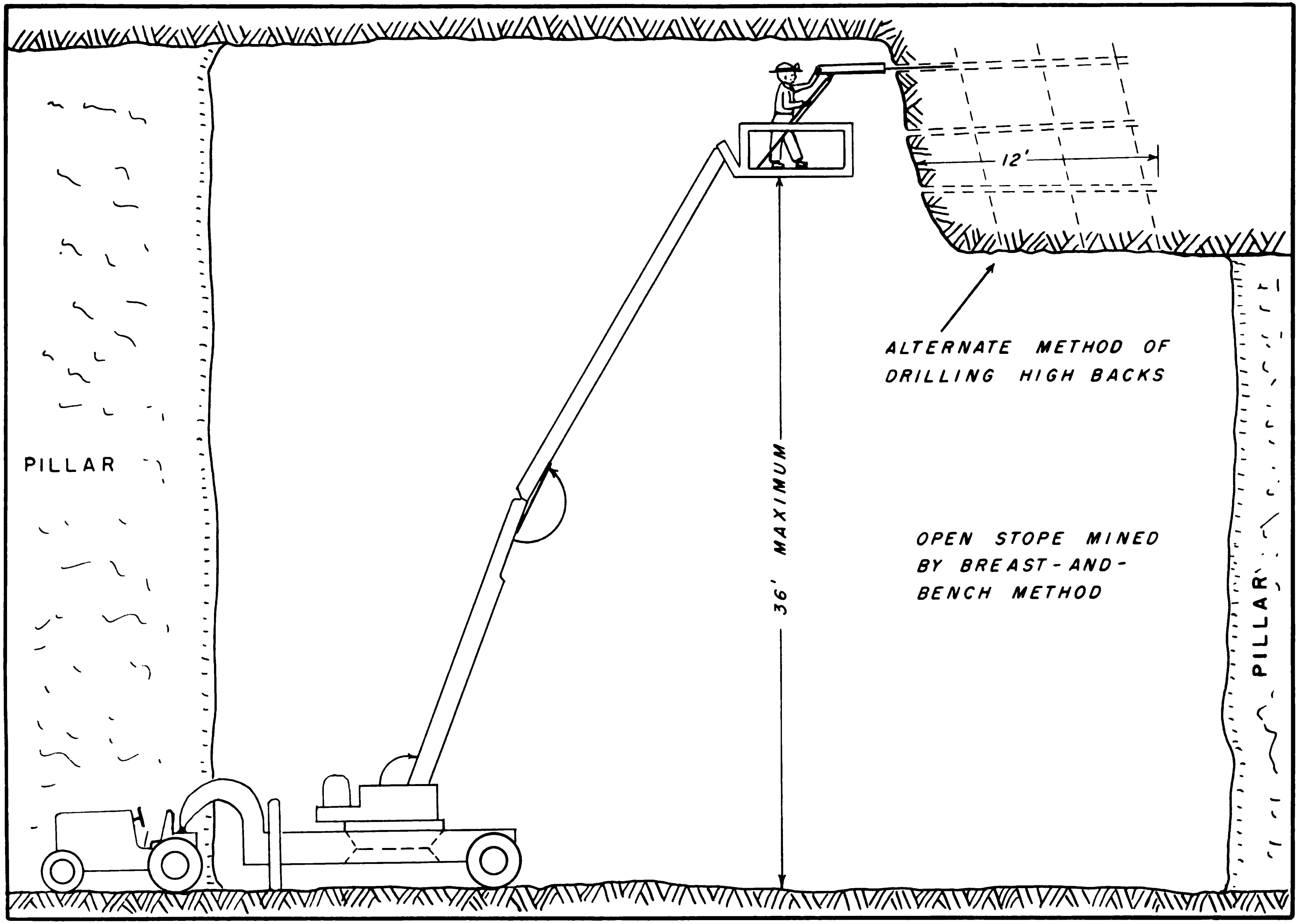
PILLAR
12'
36' MAXIMUM
ALTERNATE METHOD OF DRILLING HIGH BACKS
OPEN STOPE MINED BY BREAST-AND-BENCH METHOD
PILLAR

FIGURE 10. - Drilling for Back Slabbing.

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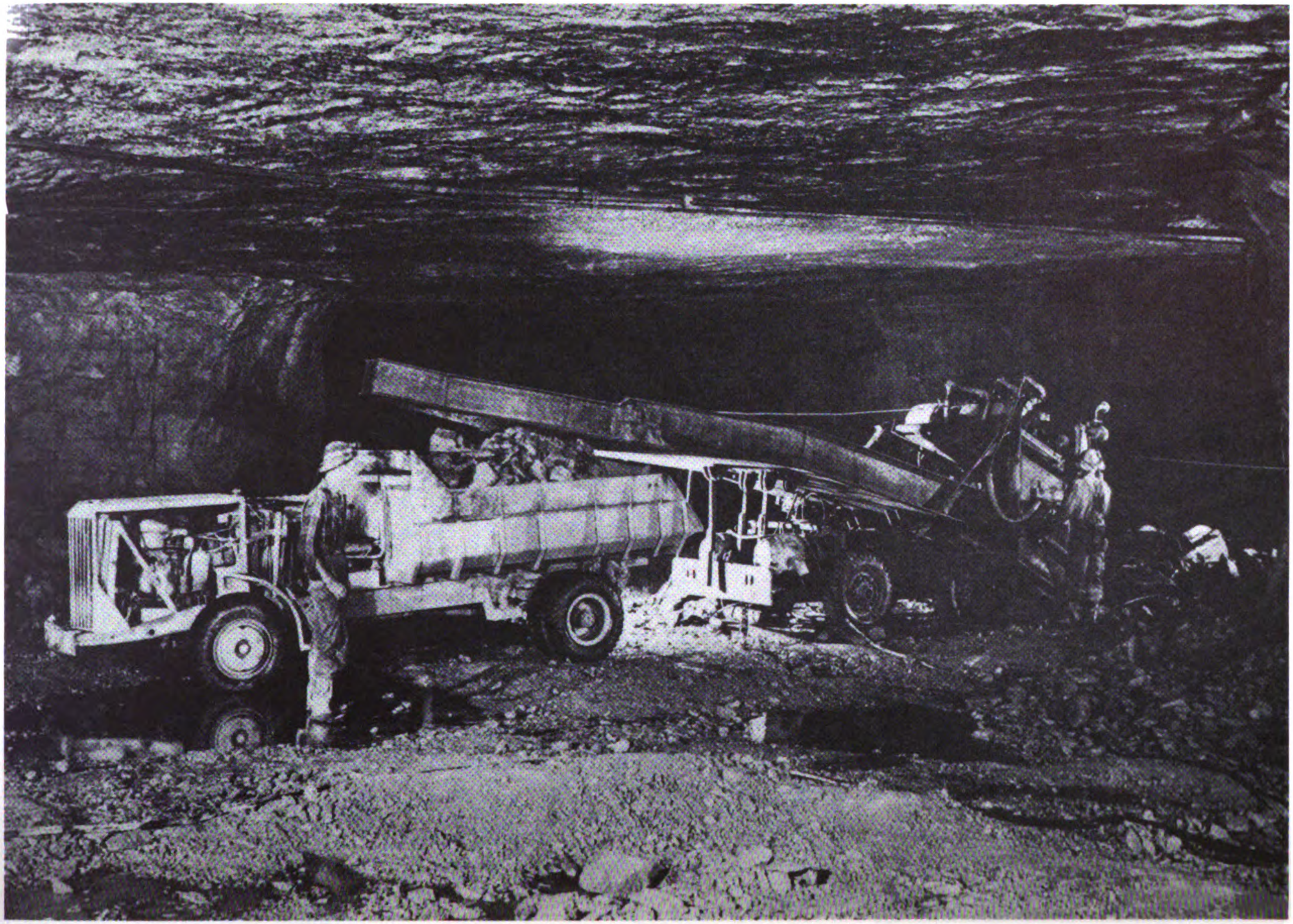

FIGURE 11. - 6-Ton Truck and Scraper-Ramp Loader.

ability to remove ore from the high, steep faces of the stopes; its ease of movement from one stope to another; and the low headroom required for its operation.

The loader is assembled on a reinforced Joy Drillmobile chassis, which has a chain drive powered by two 10-horsepower electric motors capable of moving the machine 1 mile per hour and up 20-percent grades. The loader is mounted on 8.25 by 16 tires, and the front wheels support a very heavy load. The company has determined that tire life is greatly prolonged by using solid-front tires, built of radial laminations of tire sections. The rear wheels are equipped with pneumatic tires. The ramp end of the scraper slide pivots on its support near the center of the chassis. The lip of the ramp may be raised or lowered by a pair of hydraulic jacks mounted near the front of the chassis. A 30-hp., triple-drum, electric scraper hoist with air-operated clutches is mounted above the front end of the slide. The pull rope passes back through a sheave mounted at the rear end of the slide, thence forward to the bail of the scraper.

The sides of the slide are built of structural-steel channels 15 inches wide, and the bottom is 1/2-inch steel plate. The entire trough is lined with 3/8-inch, wear-resistant, alloy steel to give a long-lived, replaceable, wearing surface. The slide is 6 feet wide, and its slope ranges from 26° at the front to 6° at the rear. The rear 8 feet of the slide is tapered and serves only as the base for the pull-rope sheave. Rollers are mounted around the lip of the slide to prevent rope wear. The scraper ramp is 28 feet 2 inches long, 7 feet 6 inches wide, and 8 feet 9 inches high.

Wire rope, 5/8 inch in diameter, is used to pull the scraper. Tail sheaves, 10 inches in diameter, are fastened to the face with "slings," flexible eyebolts consisting of a standard closed socket attached by a short length of wire rope to another socket fitted with an outside wedge. The semi-box-type scraper is 54 inches wide and has a capacity of about 0.9 ton of ore.

The unit is capable of loading 3 tons of ore per minute. In 1956 the average quantity of ore loaded per machine-shift was 224 tons.

After a face has been blasted the fly rock on the roads and stope floor is moved to the muckpile with a bulldozer. The scraper loader is driven to the pile, and the slide is jacked into the proper position for loading. After the tail sheaves have been anchored to the face, the drum cables are pulled out and placed around them. An air hose is attached to supply compressed air to the clutch control. The ore is then scraped up the slide to a hole in the rear, under which a truck or shuttle car has been driven. When the stope has been cleaned, cables and hose are disconnected and the loader can be driven or towed away. Unit loading costs in 1956 were as follows:

	Cost per ton
Operating labor	$0.086
Power	.007
Maintenance	.115
Total	.208

Haulage (3)

Mine haulage methods underwent changes as haulage distances increased. At first, Joy 12-ton shuttle cars, equipped with cable reels and electric trailing cables,were used. Hauling ore with shuttle cars proved uneconomical for distances greater than 1,000 feet, because the rate of travel was slow, not exceeding 4 miles per hour. Diesel-powered trucks gradually replaced shuttle cars as haulage distances increased. One shuttle car was converted to diesel-electric power and retained for short hauls.

The truck first selected for haulage in the Indian Creek mine is a 6-ton truck that was designed by engineers of St. Joseph Lead Co. for use in the narrow haulageways of older mines (fig. 11). Six of these trucks are still used regularly at Indian Creek. They have a top speed of 26 miles per hour and can climb 25-percent grades in low gear and 12-percent grades in second gear. Each truck is equipped with rear-wheel airbrakes, hydraulic dump, standard truck springs with overload and radius arms mounted on the rear, fluid clutch, and dual rear wheels. Front and rear tires are 12-ply, 7.50 by 15. The trucks are 18-1/2 feet long, 6-1/2 feet wide, and 4-1/2 feet high.

The 6-ton truck is used with good results, but time studies indicated that trucks of larger capacity could be used to advantage with a reduction of haulage costs. Thus, 10-ton trucks were designed, constructed, and recently put into operation. (See fig. 12.) The haulageways are wide, so the limiting dimension was the 5-1/2-foot maximum height allowable for use with the scraper loaders. These trucks are 20-1/2 feet long, 8-1/2 feet wide, and 5-1/2 feet high. Instead of having springs, the rear axle is attached to the frame by a rubber-block and steel-pin device. The front axle is mounted on sliding-pad springs and radius arms. Replaceable wearing strips are welded to the bottom of the box to prolong its life. The performance of the truck has been excellent, and at present (mid-1957) six are in use.

The change from shuttle cars to diesel trucks resulted in higher speed haulage; fewer units were required and maintenance was simpler.

Underground roads are surfaced with crushed rock and maintained with a diesel-powered road grader.

The following table compares haulage costs for the different units. (Note that the shuttle car is used only on hauls of less than 1,000 feet, whereas the trucks are used for much greater distances.) Costs shown for the 10-ton trucks are based on the first 5 months of their operation; a comprehensive cost for maintenance of the 10-ton trucks cannot be given.

Haulage costs per ton of ore

	Shuttle car	6-ton truck	10-ton truck
Operating labor	$0.080	$0.120	$0.080
Power	.010	.020	.020
Maintenance	.101	.124	
Total	.191	.264	

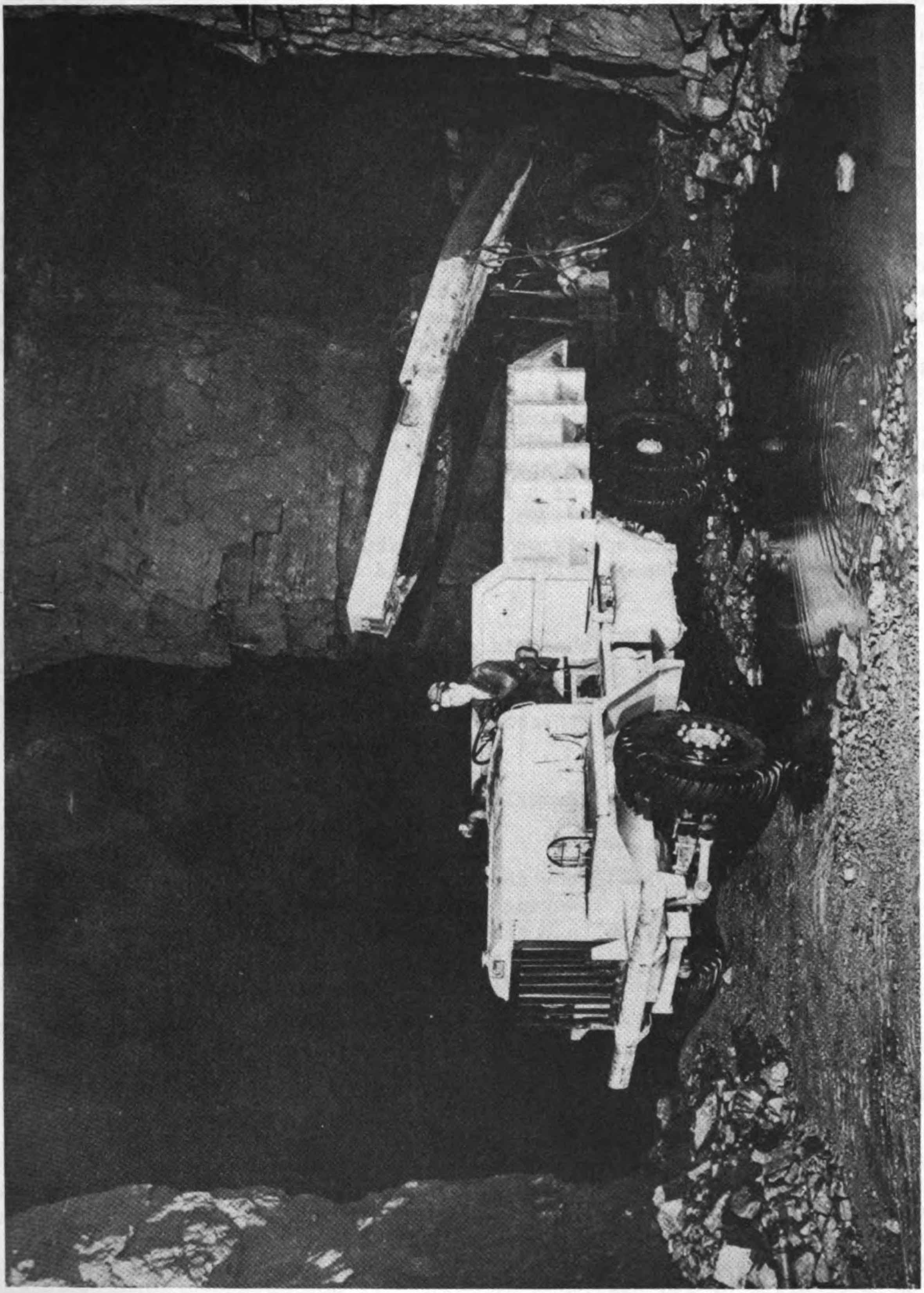

FIGURE 12. - 10-Ton Truck.

Haulage and loading crews also participate in the bonus system. Each truckdriver has a card punched by a time clock at the shaft station when he delivers a load. This enables the company to keep an accurate record of the production from each stope, also a record of the tons loaded by each loading unit and hauled by each truck.

Hoisting

Ore is hauled directly from the loader to the shaft station where it is dumped through a grizzly with 2-1/2-foot-square openings into the ore pocket, which has a capacity of 1,500 tons. An operator in the skip-loading compartment draws ore from the pocket by means of a pan feeder, which discharges into a measuring hopper. (See fig. 13.) An underswung arc gate allows the hopper to discharge into the skip. Both compartments of the shaft are used for hoisting ore, and the skip-loading facilities are identical.

The hoist has a single drum, 10 feet in diameter, on which 1-5/8-inch ropes are overwound and underwound in 1 layer. It is powered by a 600-hp., 2,300-volt motor and has a rope speed of 816 f.p.m. The skips are Kimberley type, and each has a capacity of 8 tons.

Usually ore is hoisted only on the day shift; however, a hoistman is always on duty, and ore can be hoisted as needed to fill the requirements of the mill.

Any rock spilled in the shaft is guided by deflecting plates into a spill skip at the bottom of the shaft. When the skip is full, it is attached to the bottom of the hoisting skip and elevated slightly above the measuring hopper. The contents are discharged into the hopper, and the spill skip is lowered into position.

Auxiliary Operations

Auxiliary Shaft

The concrete-lined auxiliary shaft serves as a ventilation shaft and emergency escapeway and for lowering equipment into the mine. Steel dividers and guides were omitted from this shaft. Hoisting facilities comprise a steel headframe and a 250-hp., cylindrical-drum hoist. When the hoist is not in use, the collar of the shaft is covered with a compressed-air-operated sliding door.

Trucks, shuttle cars, scraper loaders, and other equipment can be lowered through the auxiliary shaft without being dismantled.

Compressed Air

The compressor plant, which is installed in the hoistroom, consists of twin, semiradial, two-stage compressors. Each unit has four double-acting cylinders. One double-shafted, 600-hp., synchronous electric motor, mounted between the compressors, drives both of them. With this arrangement one compressor can be operated while the other is shut down for maintenance. The compressors have a total displacement of 3,200 c. f. m. and produce compressed air at a pressure of 100 p.s.i.

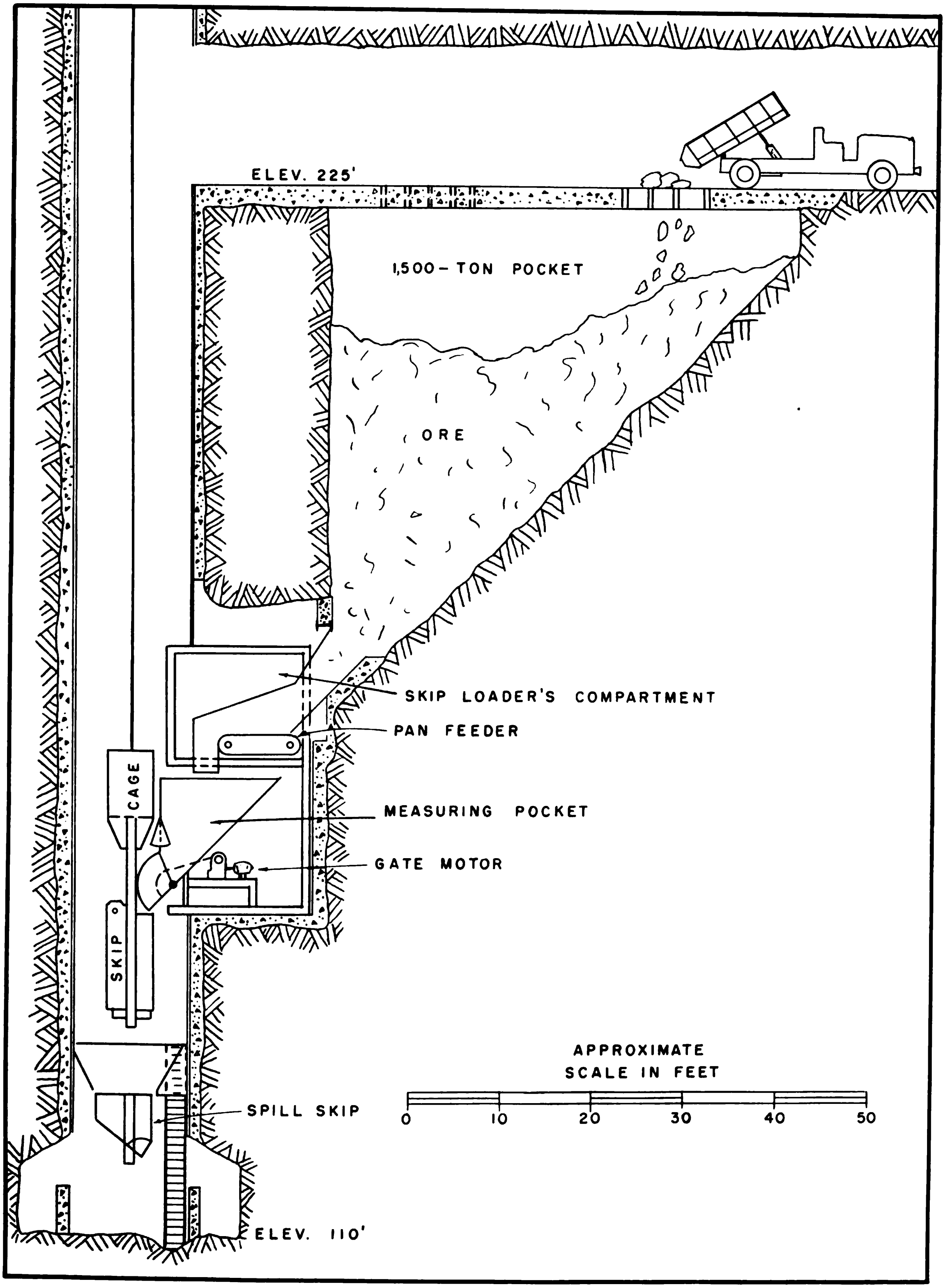

FIGURE 13. - Skip-Pocket Arrangement.

Compressed air is transmitted through a 10-inch pipeline, installed in the shaft lining, to the bottom of the main shaft, thence through 12-inch spiral-weld pipelines along the main haulageways. Three-inch lines branch off from the main lines to the stopes.

Drill-Water Supply

Water for drilling near the hoisting shaft is supplied through a line that taps the discharge column of the main pumps. Pressure to the drills is controlled by a pressure-reducing valve in the waterline. Clear drainage water is pumped to more remote areas with 1-1/2-hp. pumps.

Communications

A dial-telephone system connects all major sections of the plant and extends to the bottom of the shaft. The system includes a call bell in code, which is used occasionally to summon men who are away from their stations to the nearest telephone. The telephone line can be cleared for dialing Potosi numbers.

An intercom with loud speakers connects the hoistroom, crusherroom, and two shaft stations.

Changehouse

The changehouse was designed and built with a view to the comfort and health of the workmen. It has a radiant-heating system, with piping imbedded in the concrete floor. The floor is surfaced with antibacterial cement to prevent athlete's foot. Each workman has a locker for street clothes and a hanging basket for work clothes. The lockers are recessed into the walls, eliminating surfaces upon which miscellaneous articles and dust might accumulate. Shower stalls and wash fountains are spaced throughout the room near the lockers. Walls around the showers are low, so that the men can see their lockers while bathing.

A tunnel from the changeroom to the shaft, 20 feet below the collar, enables the mine crew to enter or leave the mine without being exposed to the weather.

Maintenance and Repair

Shops and warehouses were built and equipped at the Indian Creek plant because of its distance from St. Joseph's main operations in the Lead Belt. A surface shop is equipped for constructing and maintaining plant facilities. An underground shop near the shaft station is being equipped with a bridge crane, grease pit, and welding shop. Maintenance and repair of all underground equipment is performed here. A tool and supply room is under construction adjacent to the underground shop.

Power, Equipment, and Supplies

Power

Twelve miles of powerlines were installed to transmit electric power from the Arkansas-Missouri Power Co. lines to the transformer substation at the mine. This substation contains two 3,750-kv.-a., 3-phase, 33,000/2,300-volt transformers. Provision was made to add a third if the capacity of the mill increases. Two 2,400-volt cables carry the current to the mine. Power to the various loads is distributed from a 2,400-volt bus in a transformer station near the bottom of the shaft. All underground power cables are rubber-insulated, 3-conductor type, shielded with galvanized-wire armor. A bank of 2,400/440-volt transformers at the bottom of the shaft supplies power for the shop and for part of the mining equipment within 1,500 feet. A second bank of transformers, northeast of the shaft, serves electrically operated equipment in that part of the mine.

The scraper-ramp loaders are the chief consumers of electricity in the mine. They are powered by alternating-current motors and connected to individual circuit breakers at the power source by neoprene-covered trailing cables. The cables are 3-conductor, rubber-insulated, shielded cables with 3 ground wires in the interstices and are made in 500-foot sections with polarized plugs for connecting to other lengths of cable or to the loader. The circuit breakers protect the loader and the cable. They are equipped with ground-current relays connected through the ground wires to the frame of the machine and set to trip at a load of 10 amperes.

A 250-volt, direct-current generator, driven by a 200-hp. motor, was once used to supply power to the shuttle cars but has been eliminated.

Table 5 gives the horsepower ratings of all motors used for operating development and mining equipment.

Equipment

The major items of equipment used in operating the Indian Creek mine are listed, with numbers of each, in table 5.

Supplies

The plant warehouse carries a 90-day inventory of parts purchased from outside sources and a 30-day stock of items procured from Bonne Terre Farming & Cattle Co., a St. Joseph Lead Co. subsidiary that deals in industrial supplies.

All electric motors have been standardized to simplify repairs and interchanges and keep to a minimum the stock of necessary replacements.

TABLE 5. - Operating equipment, development, and mining

Item	Quantity	Type of power	Rated hp. (electric)
Air compressor, 3,200 c. f. m., twin-unit	1	Electric	600
Hoist, ore	1	do.	600
Hoist, material (standby, main shaft)	1	do.	150
Hoist, auxiliary-shaft	1	do.	250
Pumps, drill-water	3	do.	1-1/2
Pumps, secondary	3	do.	15
Pumps, main	2	do.	200
Fan, blower (auxiliary-shaft)	1	do.	7-1/2
Fan, auxiliary blower	1	do.	18
Do.	3	do.	3
Scraper loaders	6	do.	30
Slushers, double-drum	2	do.	30
Conveyor, waste-rock	1	do.	40
Drill jumbo	1	do.	10
Drills, air-leg	20	Air	
Drill, diamond	1	do.	
Loaders, rocker-shovel	2	do.	
Trucks, 6-ton	6	Diesel	
Trucks, 10-ton	6	do.	
Shuttle car	1	do.	
Road grader	1	do.	
Bulldozer	1	do.	
Drill platform, mobile	1	do.	

PRODUCTION RATES

About 2,000 tons of ore a day is mined on 2 shifts. In 1956 production averaged 24.36 tons per man-shift for mine employees and 16.87 tons per man-shift for all employees at Indian Creek. The respective averages for the first half of 1957 were 28.9 and 18.5 tons per man-shift.

The weighted average grade of ore mined to the end of 1957 was 2.47 percent of lead and 0.33 percent of zinc (dry basis). The lead content of the ore is lower than had been indicated by core drilling. The lower lead content is due mainly to dilution in mining and production of low-grade ore that had not been included in the original estimate. Table 6 shows the yearly output of ore, by grade.

TABLE 6. - Mine production data

Year	Ore produced, short wet tons			Average grade (dry basis)	
	By stoping	By development	Total ore hoisted	Lead, percent	Zinc, percent
1954	154,845	30,425	185,270	2.18	0.25
1955	282,669	24,796	307,465	2.55	.53
1956	455,953	23,294	479,247	2.48	.35
1957	489,445	33,343	522,788	2.52	.22
Total	1,382,912	111,858	1,494,770	2.47	.33

The normal crew employed for mining and development in mid-1957 comprised 82 men. The job classifications and number in each are as follows:

Job classification	Number of men
Mine captain	1
Mine-shift foremen	2
Mine mechanical foremen	1
Development drillers	8
Operating drillers	33
Scraper operators	8
Drivers	10
Skip loader	1
Hoist engineers	3
Underground diamond-drill operator	1
Bulldozer operators	2
Laborers	6
Maintenance men	6

The crew is divided about equally between the two operating shifts, except that 3 maintenance men work on day shift, 2 alternate between day and night shifts, and 1 works an off shift (10:30 a.m. to 6:30 p.m.). One hoist engineer is on duty each of the 3 shifts.

VENTILATION

The mine is ventilated both by natural air currents and forced air. The difference in atmospheric pressure at the collars of the main and auxiliary shafts causes an average airflow of 65,000 c. f. m. through the workings in cool weather. In hot weather a 7-1/2-hp. blower fan near the top of the auxiliary shaft is used to move the air. The fan is installed in a short drift 10 feet below the collar of the shaft, and the intake air is drawn from the surface through a short raise. This arrangement leaves the entire collar area of the shaft free for hoisting or lowering equipment. When the shaft is not in use, it is covered by steel doors to prevent loss of air pressure. Because the mine workings are open and extend only short distances to either side of the main airway, most of the stopes are well ventilated. Auxiliary blower fans deliver air through 12-inch flexible tubing to stopes or development openings that require additional ventilation.

All trucks and other diesel-powered machines are equipped with exhaust scrubbers. Samples of air are analyzed periodically to insure that dangerous conditions do not exist.

DRAINAGE

The mine pump station is about 250 feet east of the main shaft. (See fig. 3, p. 10.) The pumping plant consists of two 2-stage centrifugal pumps driven by 200-hp., 2,300-volt motors, which receive their power from the bus in the transformer station near the bottom of the shaft. Water is pumped to the surface through a 10-inch pipe installed in a churn-drill hole. The southern part of the mine serves as the sump toward which water from the north and east flows by gravity (fig. 3). Much of the water from the east, however, is diverted to the sump through a short drift that extends northward and taps the drainage of the main haulageway from the eastern section of the mine. The drift has a concrete dam on the southern end, which allows only sediment-free water to overflow into the sump (fig. 3). Pumping is required to drain the water from two low-lying stopes and shaft bottom and move it to the sump. A 15-hp. LaBour pump is used at each place. Approximately 300 to 400 g. p. m. of water flows into the mine workings. The sump has an estimated storage capacity of 800,000 gallons - enough for 33 to 45 hours of inflow.

To avoid high power demands, the main pumps are used only between shifts, when the hoist and crusher are not in operation. The water is pumped to the surface reservoir, which supplies the mill with water.

WAGE AND BONUS SYSTEM (5)

Each St. Joseph Lead Co. employee receives a lead bonus, based upon the daily market price of lead, in addition to his established daily wage. Most mine employees work on contracts and can earn additional income under an incentive bonus plan by exceeding the minimum production levels established for participation in the plan.

The following daily wage schedule was in effect on July 1, 1957:

Job classification	Authorized rate
Subshift foreman	$17.45
Development driller[1]	14.02
Operating driller[1]	14.02
Scraper operator[1]	14.02
Driver[1]	13.96
Shift extra man[1]	14.02
Skip loader	14.02
Hoist engineer	15.02
Roofman[1]	14.02
Diamond-drill operator[1]	14.02
Hammer-drill prospector[1]	14.02
Underground laborer	13.96
Bulldozer operator	14.02
Maintenance man	14.02-15.02

[1] Employees participating in incentive bonus plans.

The daily lead bonus in effect on July 1, 1957, was $1.125.

SAFETY AND HYGIENE

The mine is inspected at 2- or 3-week intervals by members of the St. Joseph Lead Co. Safety Department. A written report of the findings goes successively to the superintendent, mine captain, and shift bosses. The mine foreman is responsible for conformance to recommendations of the Safety Department and compliance with all established safety measures.

In addition to making inspections, the Safety Department also maintains bulletin boards on safety matters, provides safety equipment, and gives first-aid training. Four safety meetings on company time are held each year at the beginning of shifts. Slides or films on accidents and hazardous conditions are shown at these meetings, and discussions are held. Occasionally, additional meetings on safety are held by the shift bosses.

All personnel employed underground is required to wear hard hats, safety-toed shoes, and safety glasses. Wearing safety glasses was made mandatory because of the comparatively high incidence of minor injuries to eyes. Table 7 gives accident statistics for the mine.

TABLE 7. - Accident statistics[1]

	1951		1952		1953		1954	
	Number		Number		Number		Number	
	LT	NLT	LT	NLT	LT	NLT	LT	NLT
Shaft sinking								
Falling objects				1		2		2
Loading rock				3		4		1
Handtools		3				4		
Slips and falls		2		2		4		1
Drilling		1		5		4		1
Machinery				3		4		2
Material handling						2		2
Bucket or cage		1		2				1
Burns		3				1		
Other		5		8		3		5
Total		15		24		28		15
Development and mining								
	1953		1954		1955		1956	
Falling objects		1			1(1)	2		1
Loading rock		2		5	1(26)	7		2
Handtools		1		4		1		1
Slips and falls	1(24)	2	1(7)		1(10)	8	1(194)	5
Drilling		1		4	4(47)	15	6(62)	5
Machinery		1						
Material handling						1		3
Hauling		1		1			1(4)	3
Burns				1				1
Other	1(75)	2	1(25)	7		6		5
Total	2(99)	11	2(32)	22	7(84)	40	8(260)	26

[1] LT = lost-time injuries; NLT = no-lost-time injuries; figures in parentheses indicate number of shifts lost.

A first-aid room next to the changehouse is well equipped with supplies essential to the care of wounds and fractures; it has a cot, a special wash-basin, and an inhalator. A member of the staff, trained in first-aid work, is always available for assisting an injured person.

MILLING OPERATIONS

Ore is hoisted from the mine on day shift only, at a maximum rate of 2,000 tons in 6-1/2 hours. The mill is operated 24 hours a day, 5 days a week.

The crew comprises a mill foreman, 10 men on shift 1, and 2 men each on shifts 2 and 3. The duties of shift 1, in addition to milling the ore, include preparing samples; receiving and crushing run-of-mine ore for the day; drying, loading, and hauling the concentrates that were produced the day before; and doing maintenance work. The duties of shifts 2 and 3 are concerned strictly with operating.

Figure 14 is an exterior view of the mill building and the headframe of the hoisting shaft. Figures 15, 16, and 17 are detailed flowsheets of various sections of the concentration plant.

Crushing

Mine-run ore is dumped by skip into a 50-ton steel receiving bin, equipped with a 58- by 56-inch, drop-bar, feeder-type grizzly with 3-inch openings. The oversize is crushed in a Telsmith 25-B gyratory crusher set at 3 inches. The crushed ore and grizzly undersize are combined and conveyed to two 5- by 8-foot, single-deck, Nordberg screens having 3/4- by 6-inch screen openings. The screen oversize is crushed in two Allis-Chalmers 460 Hydrocone crushers, each of which is in closed circuit with a Nordberg screen. The screen undersize is weighed by a Merrick weightometer and conveyed to a 2,200-ton steel storage bin. The conveyor is equipped with an automatic tripper for ore distribution. The crushed ore is withdrawn beneath the bin to six conveyors, which discharge onto a central conveyor. The ore is then weighed by another Merrick weightometer and transported on the central conveyor to the rodmill and jig circuit.

Two optional circuits are available for handling waste rock. In one circuit, chutes are placed so that rock dumped from the skip is diverted into a 120-ton steel bin, from which it is withdrawn into trucks for disposal. In the other circuit the waste rock passes through the crushers and screens to the conveyor above the 2,200-ton ore bin, whence it is diverted by a removable chute to a stockpile outside the mill building.

Grinding and Jigging

The crushed ore is ground in open circuit at 70 percent solids in a 9- by 12-foot Allis-Chalmers rodmill, loaded with 96,000 pounds of 3-inch-diameter steel rods. This operation reduces the ore to a pulp consisting of 39 percent on 20-mesh, 41 percent on 100-mesh, and 20 percent through 100-mesh. The rods, which contain 0.75 percent carbon, 0.75 percent manganese, 0.05 percent phosphorus, and 0.05 percent sulfur, are consumed at a rate of 0.1779 pound per ton of ore ground. Aerofloat 31, the function of which is explained later in this report, is added to the rodmill feed at the rate of 0.066 pound per ton of feed.

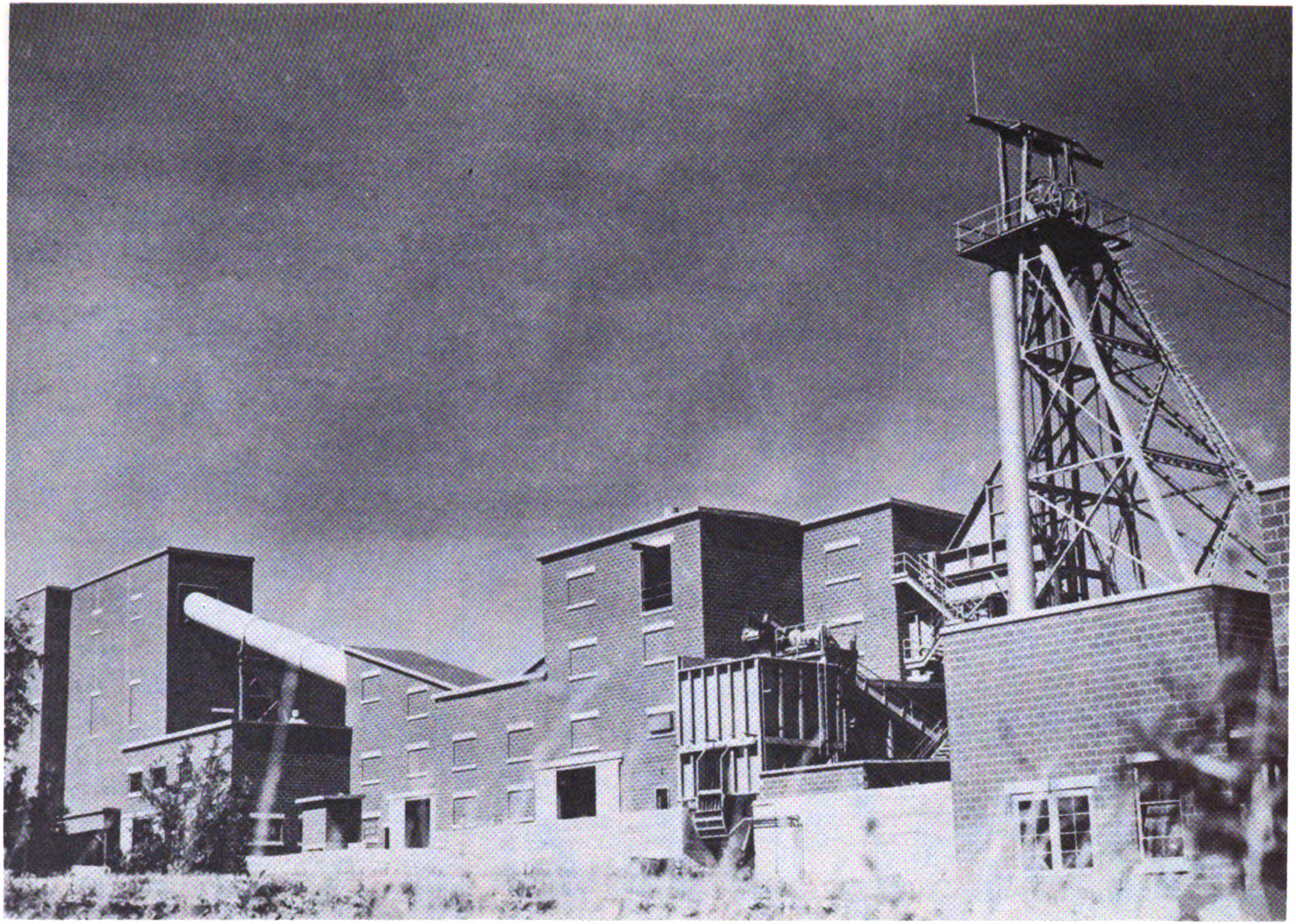

FIGURE 14. - Mill Building.

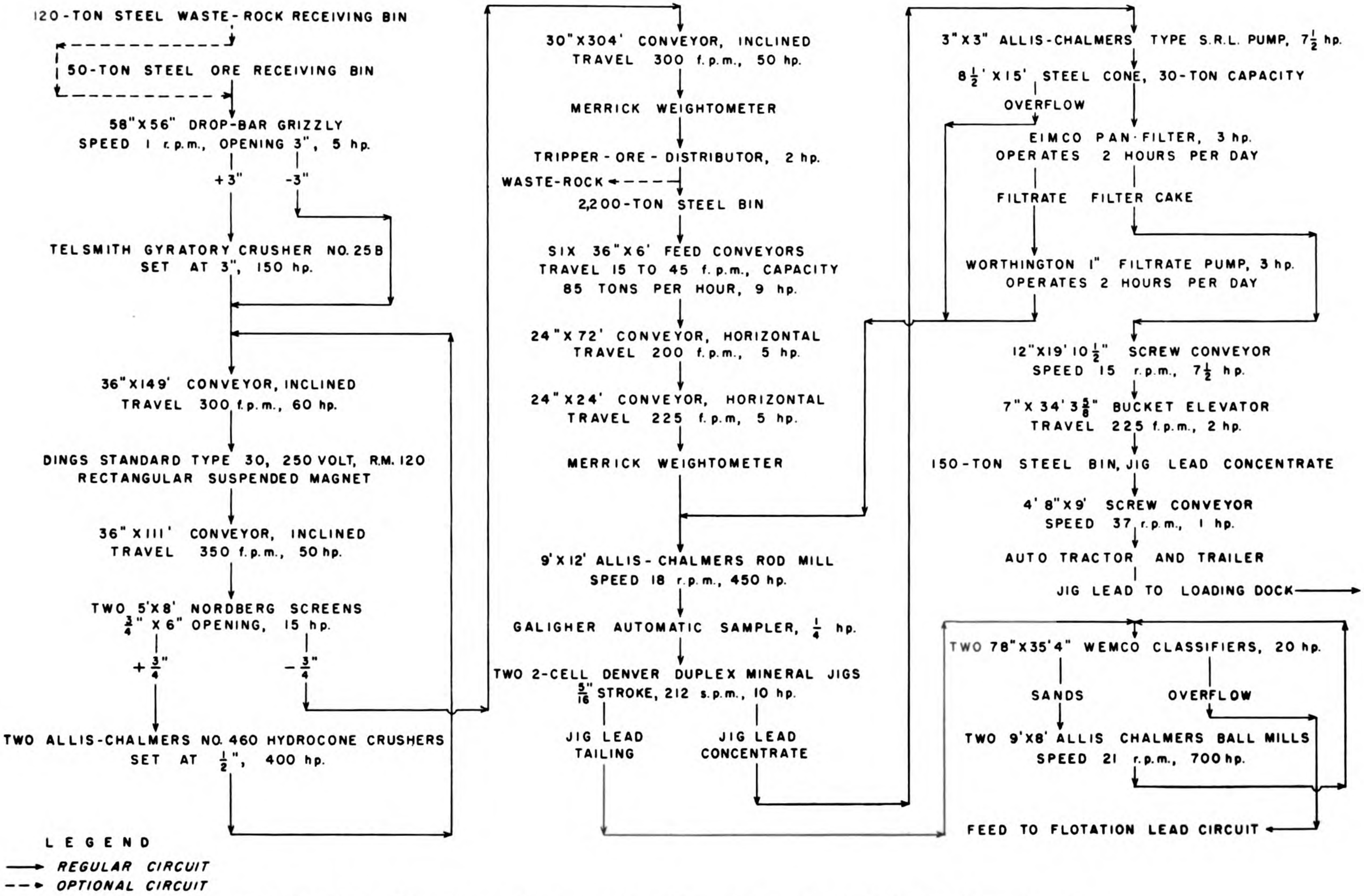
120-TON STEEL WASTE-ROCK RECEIVING BIN
50-TON STEEL ORE RECEIVING BIN
58"X56" DROP-BAR GRIZZLY
SPEED 1 r.p.m., OPENING 3", 5 hp.
+3"
-3"
TELSMITH GYRATORY CRUSHER NO. 25B
SET AT 3", 150 hp.
36"X149' CONVEYOR, INCLINED
TRAVEL 300 f.p.m., 60 hp.
DINGS STANDARD TYPE 30, 250 VOLT, R.M. 120
RECTANGULAR SUSPENDED MAGNET
36"X111' CONVEYOR, INCLINED
TRAVEL 350 f.p.m., 50 hp.
TWO 5'X8' NORDBERG SCREENS
¾" X 6" OPENING, 15 hp.
+¾"
-¾"
TWO ALLIS-CHALMERS NO. 460 HYDROCONE CRUSHERS
SET AT ½", 400 hp.
30"X304' CONVEYOR, INCLINED
TRAVEL 300 f.p.m., 50 hp.
MERRICK WEIGHTOMETER
TRIPPER-ORE-DISTRIBUTOR, 2 hp.
WASTE-ROCK
2,200-TON STEEL BIN
SIX 36"X6' FEED CONVEYORS
TRAVEL 15 TO 45 f.p.m., CAPACITY
85 TONS PER HOUR, 9 hp.
24"X72' CONVEYOR, HORIZONTAL
TRAVEL 200 f.p.m., 5 hp.
24"X24' CONVEYOR, HORIZONTAL
TRAVEL 225 f.p.m, 5 hp.
MERRICK WEIGHTOMETER
9'X12' ALLIS-CHALMERS ROD MILL
SPEED 18 r.p.m., 450 hp.
GALIGHER AUTOMATIC SAMPLER, ¼ hp.
TWO 2-CELL DENVER DUPLEX MINERAL JIGS
5/16" STROKE, 212 s.p.m., 10 hp.
JIG LEAD TAILING
JIG LEAD CONCENTRATE
3"X3" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7½ hp.
8½'X15' STEEL CONE, 30-TON CAPACITY
OVERFLOW
EIMCO PAN-FILTER, 3 hp.
OPERATES 2 HOURS PER DAY
FILTRATE
FILTER CAKE
WORTHINGTON 1" FILTRATE PUMP, 3 hp.
OPERATES 2 HOURS PER DAY
12"X19'10½" SCREW CONVEYOR
SPEED 15 r.p.m., 7½ hp.
7"X34'3⅝" BUCKET ELEVATOR
TRAVEL 225 f.p.m., 2 hp.
150-TON STEEL BIN, JIG LEAD CONCENTRATE
4'8"X9' SCREW CONVEYOR
SPEED 37 r.p.m., 1 hp.
AUTO TRACTOR AND TRAILER
JIG LEAD TO LOADING DOCK
TWO 78"X35'4" WEMCO CLASSIFIERS, 20 hp.
SANDS
OVERFLOW
TWO 9'X8' ALLIS CHALMERS BALL MILLS
SPEED 21 r.p.m., 700 hp.
FEED TO FLOTATION LEAD CIRCUIT
LEGEND
REGULAR CIRCUIT
OPTIONAL CIRCUIT

FIGURE 15. - Flowsheet, St. Joseph Lead Co., Indian Creek Mill—Crushing, Jigging, and Fine-Grinding Circuit.

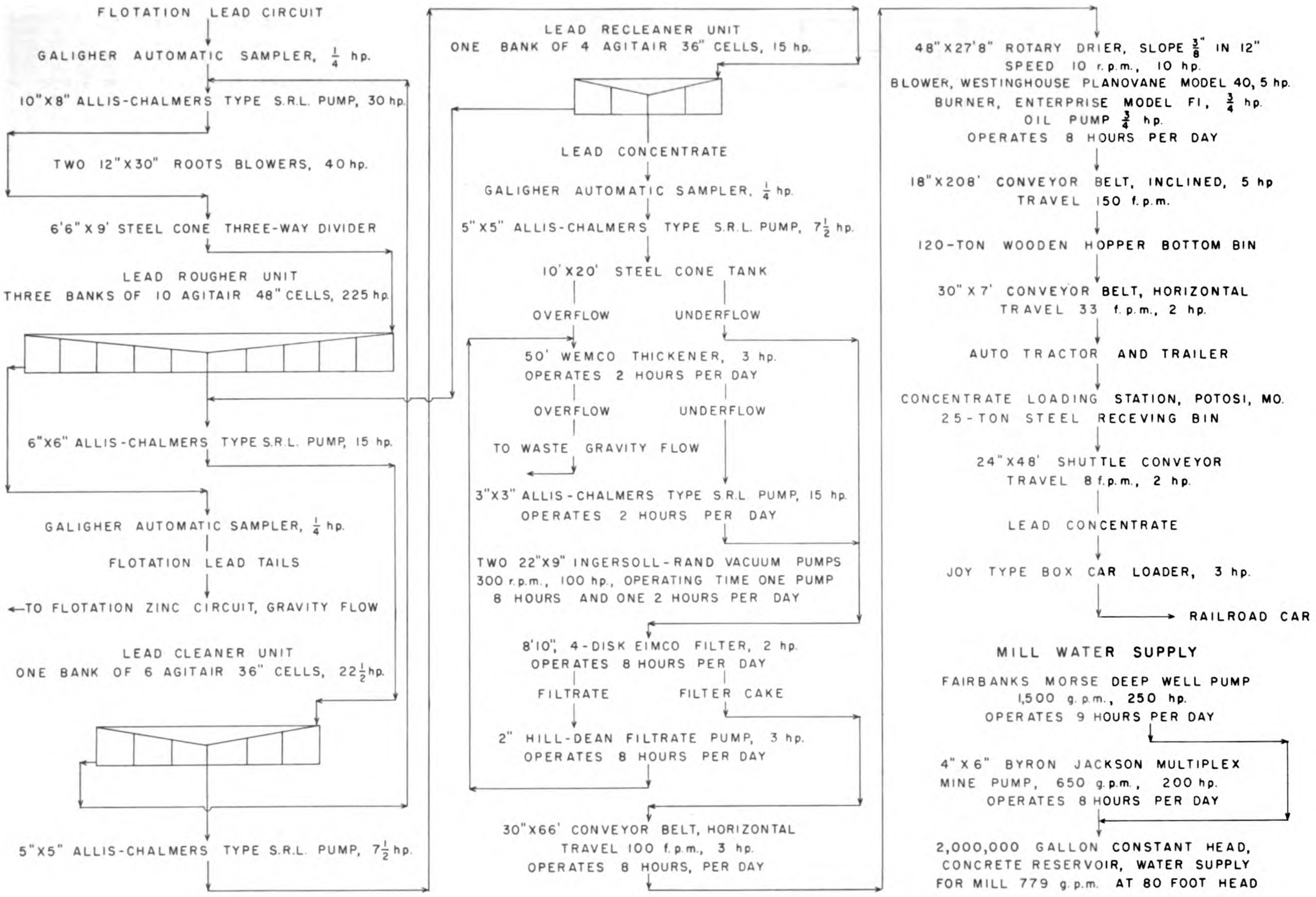
FLOTATION LEAD CIRCUIT
GALIGHER AUTOMATIC SAMPLER, 1/4 hp.
10"X8" ALLIS-CHALMERS TYPE S.R.L. PUMP, 30 hp.
TWO 12"X30" ROOTS BLOWERS, 40 hp.
6'6"X9' STEEL CONE THREE-WAY DIVIDER
LEAD ROUGHER UNIT
THREE BANKS OF 10 AGITAIR 48" CELLS, 225 hp.
6"X6" ALLIS-CHALMERS TYPE S.R.L. PUMP, 15 hp.
GALIGHER AUTOMATIC SAMPLER, 1/4 hp.
FLOTATION LEAD TAILS
←TO FLOTATION ZINC CIRCUIT, GRAVITY FLOW
LEAD CLEANER UNIT
ONE BANK OF 6 AGITAIR 36" CELLS, 22 1/2 hp.
5"X5" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7 1/2 hp.
LEAD RECLEANER UNIT
ONE BANK OF 4 AGITAIR 36" CELLS, 15 hp.
LEAD CONCENTRATE
GALIGHER AUTOMATIC SAMPLER, 1/4 hp.
5"X5" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7 1/2 hp.
10'X20' STEEL CONE TANK
OVERFLOW
UNDERFLOW
50' WEMCO THICKENER, 3 hp.
OPERATES 2 HOURS PER DAY
OVERFLOW
UNDERFLOW
TO WASTE
GRAVITY FLOW
3"X3" ALLIS-CHALMERS TYPE S.R.L. PUMP, 15 hp.
OPERATES 2 HOURS PER DAY
TWO 22"X9" INGERSOLL-RAND VACUUM PUMPS
300 r.p.m., 100 hp., OPERATING TIME ONE PUMP
8 HOURS AND ONE 2 HOURS PER DAY
8'10", 4-DISK EIMCO FILTER, 2 hp.
OPERATES 8 HOURS PER DAY
FILTRATE
FILTER CAKE
2" HILL-DEAN FILTRATE PUMP, 3 hp.
OPERATES 8 HOURS PER DAY
30"X66' CONVEYOR BELT, HORIZONTAL
TRAVEL 100 f.p.m., 3 hp.
OPERATES 8 HOURS, PER DAY
48"X27'8" ROTARY DRIER, SLOPE 3/8" IN 12"
SPEED 10 r.p.m., 10 hp.
BLOWER, WESTINGHOUSE PLANOVANE MODEL 40, 5 hp.
BURNER, ENTERPRISE MODEL F1, 3/4 hp.
OIL PUMP 3/4 hp.
OPERATES 8 HOURS PER DAY
18"X208' CONVEYOR BELT, INCLINED, 5 hp
TRAVEL 150 f.p.m.
120-TON WOODEN HOPPER BOTTOM BIN
30"X7' CONVEYOR BELT, HORIZONTAL
TRAVEL 33 f.p.m., 2 hp.
AUTO TRACTOR AND TRAILER
CONCENTRATE LOADING STATION, POTOSI, MO.
25-TON STEEL RECEVING BIN
24"X48' SHUTTLE CONVEYOR
TRAVEL 8 f.p.m., 2 hp.
LEAD CONCENTRATE
JOY TYPE BOX CAR LOADER, 3 hp.
RAILROAD CAR
MILL WATER SUPPLY
FAIRBANKS MORSE DEEP WELL PUMP
1,500 g.p.m., 250 hp.
OPERATES 9 HOURS PER DAY
4"X6" BYRON JACKSON MULTIPLEX
MINE PUMP, 650 g.p.m., 200 hp.
OPERATES 8 HOURS PER DAY
2,000,000 GALLON CONSTANT HEAD,
CONCRETE RESERVOIR, WATER SUPPLY
FOR MILL 779 g.p.m. AT 80 FOOT HEAD

FIGURE 16. - Flowsheet, St. Joseph Lead Co., Indian Creek Mill–Lead Circuit.

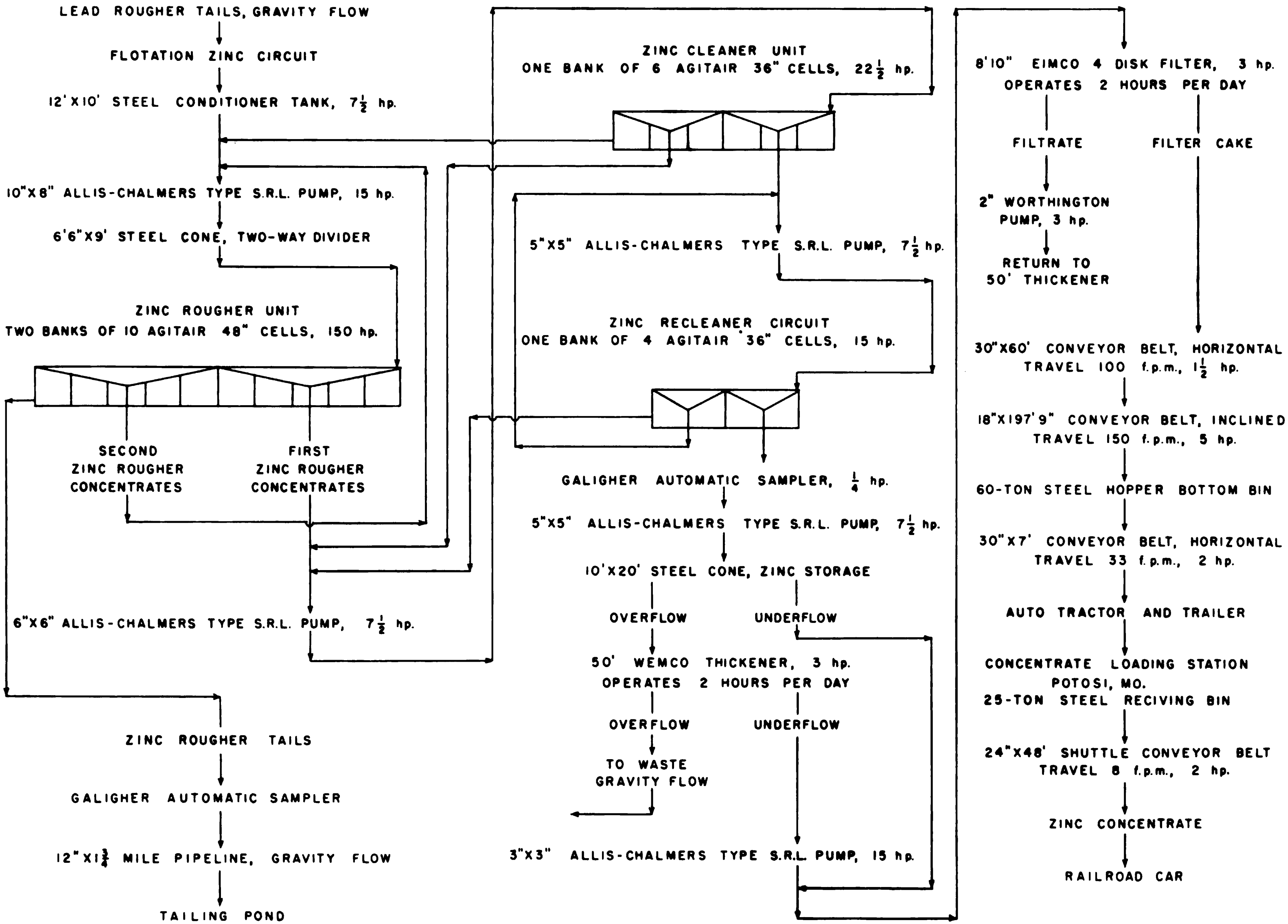
LEAD ROUGHER TAILS, GRAVITY FLOW
FLOTATION ZINC CIRCUIT
12'X10' STEEL CONDITIONER TANK, 7½ hp.
10"X8" ALLIS-CHALMERS TYPE S.R.L. PUMP, 15 hp.
6'6"X9' STEEL CONE, TWO-WAY DIVIDER
ZINC ROUGHER UNIT
TWO BANKS OF 10 AGITAIR 48" CELLS, 150 hp.
SECOND ZINC ROUGHER CONCENTRATES
FIRST ZINC ROUGHER CONCENTRATES
6"X6" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7½ hp.
ZINC ROUGHER TAILS
GALIGHER AUTOMATIC SAMPLER
12"X1¾ MILE PIPELINE, GRAVITY FLOW
TAILING POND
ZINC CLEANER UNIT
ONE BANK OF 6 AGITAIR 36" CELLS, 22½ hp.
5"X5" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7½ hp.
ZINC RECLEANER CIRCUIT
ONE BANK OF 4 AGITAIR 36" CELLS, 15 hp.
GALIGHER AUTOMATIC SAMPLER, ¼ hp.
5"X5" ALLIS-CHALMERS TYPE S.R.L. PUMP, 7½ hp.
10'X20' STEEL CONE, ZINC STORAGE
OVERFLOW
UNDERFLOW
50' WEMCO THICKENER, 3 hp.
OPERATES 2 HOURS PER DAY
OVERFLOW
UNDERFLOW
TO WASTE
GRAVITY FLOW
3"X3" ALLIS-CHALMERS TYPE S.R.L. PUMP, 15 hp.
8'10" EIMCO 4 DISK FILTER, 3 hp.
OPERATES 2 HOURS PER DAY
FILTRATE
FILTER CAKE
2" WORTHINGTON PUMP, 3 hp.
RETURN TO 50' THICKENER
30"X60' CONVEYOR BELT, HORIZONTAL
TRAVEL 100 f.p.m., 1½ hp.
18"X197'9" CONVEYOR BELT, INCLINED
TRAVEL 150 f.p.m., 5 hp.
60-TON STEEL HOPPER BOTTOM BIN
30"X7' CONVEYOR BELT, HORIZONTAL
TRAVEL 33 f.p.m., 2 hp.
AUTO TRACTOR AND TRAILER
CONCENTRATE LOADING STATION
POTOSI, MO.
25-TON STEEL RECIVING BIN
24"X48' SHUTTLE CONVEYOR BELT
TRAVEL 8 f.p.m., 2 hp.
ZINC CONCENTRATE
RAILROAD CAR

FIGURE 17. - Flowsheet, St. Joseph Lead Co., Indian Creek Mill–Zinc Circuit.

The rodmill discharge is sampled and flows by gravity to two 2-cell, Denver, duplex mineral jigs with 36- by 48-inch cell areas. The first cell of each unit is equipped with a wedge-type, 4-mm.-opening, bar-type grid, on which 1/4-inch steel shot are bedded to a depth of 1-1/2 inches. The second cell of each unit has 3-mm. openings and a bedding of 3/16-inch shot to a depth of 1-1/2 inches.

The jigs are operated at a speed of 212 strokes per minute. The length of stroke is 5/16 inch. The only finished product is a hutch lead concentrate, which is stored in an 8-1/2- by 15-foot conical steel tank. Concentrates are stored in this tank during the second and third shifts and withdrawn during the first shift into an Eimco pan-type filter. The filtrate and the overflow from the storage tank are pumped to the rodmill grinding circuit. The filtered lead concentrate is conveyed by a 12-inch by 19-foot 10-1/2-inch screw to a 7-inch by 34-foot 3-5/8-inch bucket elevator.

The elevator discharges into a 150-ton steel storage bin equipped with a 4-foot 8-inch by 9-foot screw conveyor. The conveyor loads concentrates into an "auto" tractor and trailer of 38,000 pounds capacity for transportation to the railroad loading platform at Potosi, Mo.

Fine Grinding and Lead Flotation

The tailings from the two jigs are ground separately in two 9- by 8-foot Allis-Chalmers ballmills in closed circuit with two 78-inch by 35-foot 4-inch Wemco classifiers. Each ballmill is charged with 55,000 pounds of 2- and 1-1/2-inch steel balls. The ore is ground at a dilution of about 70 percent solids. The ground feed averages 6 percent on 48-mesh, 11.5 percent on 65-mesh, 11.5 percent on 100-mesh, and 71 percent through 100-mesh. The balls, which contain 0.7 percent carbon, 0.3 percent molybdenum, and 0.25 percent copper, are consumed at the rate of 0.145 pound per ton of ore ground.

The overflow from the Wemco classifiers is sampled; conditioned with 0.38 pound of NaCN, 0.065 pound of $ZnSO_4$, and 0.046 pound of No. 77 Aerofroth per ton of feed; then pumped to a 6-1/2- by 9-foot, 3-way, conical steel divider, which splits the feed to 3 banks of ten 48-inch Agitair lead rougher cells.

Air for all flotation cells is supplied by two 12- by 30-inch Roots blowers.

The rougher lead concentrate is pumped to a cleaner unit comprising a bank of six 36-inch Agitair cells. The cleaner lead concentrate is pumped to a recleaner unit consisting of 1 bank of four 36-inch Agitair cells.

The lead recleaner concentrate is sampled and pumped to a 10- by 20-foot, conical, steel dewatering tank, from which the overflow goes to a 50-foot Wemco thickener. The underflow from the steel dewatering tank flows by gravity, and the thickener underflow is pumped to an 8-foot 10-inch Eimco 4-disk filter. The overflow from the Wemco thickener flows to waste. The filtrate from the filter is returned to the thickener. Vacuum for filtration is supplied by two Ingersoll-Rand 22- by 9-inch vacuum pumps.

The filtered lead concentrate is conveyed to a 48-inch by 27-foot 8-inch, oil-fired, rotary drier, thence to a 120-ton, hopper-bottom, wood storage bin. It is withdrawn from this bin by a conveyor-belt-type feeder and delivered in a 38,000-pound-capacity "auto" tractor and trailer to a 50-ton steel receiving bin at the railroad loading station. The lead concentrate is drawn from this bin, conveyed by a shuttle conveyor, and loaded into railroad cars by a Joy boxcar loader. All lead concentrates are shipped to the Herculaneum (Mo.) smelter.

Zinc Flotation

The rougher flotation tailing is sampled; then, after 0.12 pound of copper sulfate per ton has been added, the tailing is pumped to a 12- by 10-foot steel conditioning tank. The conditioned tailing is pumped to a 6-1/2- by 9-foot, steel, 2-way conical divider. Here the product is split into 2 banks of ten 48-inch Agitair flotation cells, which constitute the zinc rougher circuit. The low zinc content of the ore has necessitated a unique system of recirculation of products in this circuit to control the grade of concentrate.

The rougher concentrate from the last 5 cells in both banks is returned to the rougher feed; the tailing, after it has been sampled, flows by gravity to the central entrace of the tailing-disposal line. The rougher concentrate from the first 5 cells of each bank is pumped to a cleaner unit consisting of 1 bank of six 36-inch Agitair cells. The concentrate from the last 3 cells is recirculated to the cleaner feed; tailings from the last 3 cells are returned to the rougher feed. The concentrate from the first 3 cells is pumped to the recleaner circuit - 1 bank of four 36-inch Agitair cells. The concentrate from the last 2 cells of the recleaner circuit is returned to the recleaner feed; tailings from the last 2 cells join the first cleaner feed. The first 2 cells of the recleaner unit produce the final zinc concentrate, which is sampled and pumped to a 10- by 20-foot conical steel storage tank. The overflow from this tank goes by gravity to a 50-foot Wemco thickener, the overflow from which flows to waste. The underflows from the storage tank and thickener are dewatered in an 8-foot 10-inch, Eimco, 4-disk filter, and the filtrate is returned to the thickener. The filtered concentrate is conveyed to a 60-ton, steel, hopper-bottom bin. Filtering and conveying are completed in about 2 hours a day. Zinc concentrate is drawn from the storage bin by a conveyor-type feeder into a 38,000-pound-capacity "auto" tractor and trailer and transported to a 25-ton steel receiving bin at the railroad loading platform at Potosi, Mo. From the receiving bin the concentrates are loaded into railroad cars with a shuttle conveyor. All zinc concentrates are shipped to the Josephtown (Pa.) smelter.

Tailing Disposal, Water Supply, and Ventilation

The tailing and waste water flow by gravity from the mill 1,500 feet through a 12-inch pipeline to the upper end of a tailing pond about 2 miles long.

The water supply for the mill is obtained from Fourche-à-Renault Creek and from the mine. Water is pumped from both sources to a 2-million-gallon concrete reservoir located to maintain a constant head of 80 feet at the mill. The calculated demand for the mill is 779 g. p. m.

Water is pumped from the creek with a Fairbanks-Morse deep-well pump through a 13-1/4-inch, i.-d. pipeline 3 miles long. The pump is operated 9 hours a day and delivers water to the reservoir at a rate of 1,500 g. p. m. under a 507-foot head. Water is pumped from the mine with a 4- by 6-inch Byron Jackson multiplex pump, operating 8 hours a day, under a 1,000-foot head. Water is delivered by the pump to the reservoir at a rate of 650 g. p. m. The pumps are controlled automatically and timed to run during the hours of off-peak load. Potable water is obtained from a deep well at the plant site.

All dry-crushing units, ore bins, and screens are equipped with adequate dust collectors. Four power-distribution centers - 1 in the hoistroom and 1 each for the dry-crushing, wet-grinding, and flotation circuits - are protected from dust by pressurized, filtered air. The filtered air is supplied by 4 American Blower Lineflow fans, each of which produces an air flow of 10,000 c. f. m.

Metallurgical Data

During 1956 the mill operated 255 days, treating 464,392.3 short dry tons of ore having an average analysis of 2.48 percent lead and 0.35 percent zinc. The mill produced 14,138.4 short dry tons of lead concentrate containing 79.48 percent lead and 2,505.6 short dry tons of zinc concentrate containing 57.92 percent zinc. The mill tailings contained 0.06 percent lead and 0.04 percent zinc. Data indicated a metallurgical recovery of 97.57 percent of the lead and 89.29 percent of the zinc.

A substantial reduction in lead losses was realized by changing the point of addition of Aerofloat 31 (0.066 pound per ton) from the flotation cells to the rodmill feed. Observations disclosed that Aerofloat caused considerable skin flotation of fine lead on the surface of the rodmill pulp. It was assumed that the same phenomenon would take place in subsequent ballmill grinding, so the change was made to minimize the possibility of overgrinding and losing fine galena in the flotation tailings.

MINING AND MILLING COSTS

Table 8 summarizes costs of shaft sinking, development, and mining in units of labor, power and supplies.

Tables 9 through 11 show the distribution of labor, power, and supply costs charged to milling, and table 12 summarizes milling in units of labor, power, and supplies.

TABLE 8. - Summary of costs in units of labor, power, and supplies

Shaft sinking

	Main	Auxiliary
Depth excavated, feet	948	695
(A) Labor (man-hours per foot excavated):		
Drilling	5.70	3.64
Charging, blasting, ventilation	2.64	2.18
Mucking, hand and machine	14.30	6.68
Placing forms	8.80	4.93
Concreting	5.10	2.89
Grouting	8.33	3.60
Drainage (cutting pump stations, pump installation)	4.51	1.93
Rock disposal	1.10	3.27
Miscellaneous	11.52	3.48
Total labor	62.00	32.50
Labor, percent of total cost	48.5	47.3
(B) Power and supplies:		
Explosives, pounds per foot	36.9	38.5
Concrete lining, per foot of shaft concreted excluding 10-foot collar:		
Cement, bags	14.1	15.1
Gravel, cubic yards	2.15	2.31
Sand, cubic yards	1.45	1.60
Cement for grouting, total bags	5,400	2,621
Total power, kilowatt-hours per foot	2,157	2,120
Power and supplies, percent of total cost	51.5	52.7

Development

Period worked: 1956
Lineal advance: 1,767 feet

(A) Labor (man-hours per foot):		
Supervision		0.49
Drilling and blasting		4.13
Loading		1.40
Hauling		1.16
Hoisting		.28
General (including maintenance)		2.50
Total labor		9.96
Labor, percent of total cost		48.3
(B) Power and supplies:		
Explosives, pounds per foot		17.7
Total power, kilowatt-hours per foot		131
Drilling	49	
Loading	5	
Hoisting	15	
Drainage	17	
Ventilation	8	
Lighting	7	
Miscellaneous	30	
Power and supplies, percent of total cost		51.7

TABLE 8. - Summary of costs in units of labor, power, and supplies (Con.)

Mining

Period covered: 1956
Mining method: Open stoping with casual pillar support
Tons of ore mined and hoisted: 479,247

(A) Labor (man-hours per ton):		
Supervision		0.025
Drilling and blasting		.140
Loading		.038
Hauling		.044
Hoisting		.005
General (including maintenance)		.076
Total labor		.328
Average tons per mine man-shift		24.36
Labor, percent of total cost		56.0
(B) Power and supplies:		
Explosives, pounds per ton		.39
Total power, kilowatt-hours per ton		6.1
Drilling	1.8	
Loading	.4	
Hoisting	1.1	
Drainage	1.2	
Ventilation	.4	
Lighting	.2	
Miscellaneous	1.0	
Power and supplies, percent of total cost		44.0

TABLE 9. - Distribution of total labor, power, and supply costs charged to milling, 1956

Item	Labor, percent of total cost	Power and supplies, percent of total cost	Percent of total cost
Operation	16.39	54.71	71.10
Maintenance	7.30	16.08	23.38
Supervision	5.52	-	5.52
Total	29.21	70.79	100.00

TABLE 10. - Distribution of operating labor, power, and supply costs charged to milling, 1956

Item	Operating labor, percent of total cost	Operating power and supplies, percent of total cost
Dry crushing	3.23	3.76
Fine grinding	5.60	18.46
Jigging	-	.83
Flotation	7.56	25.35
Mineral loading and chat disposal	-	.02
Water supply	-	3.37
Mill heating	-	1.22
Miscellaneous	-	1.70
Total	16.39	54.71

TABLE 11. - Distribution of mill maintenance costs, 1956

Item	Maintenance labor, percent of total cost	Maintenance power and supplies, percent of total cost
Dry crushing	3.09	4.18
Fine grinding	1.26	8.71
Jigging	-	-
Flotation	1.91	2.89
Mineral loading and chat disposal	.12	.30
Water supply	.15	-
Mill heating	-	-
Miscellaneous	.77	-
Total	7.30	16.08

TABLE 12. - Summary of milling costs in units of labor, power, and supplies

Period covered: 1956
Milling method: Gravity and flotation
Tons milled: 478,756 (wet)

	Percent of total	Man-hours per ton
(A) Labor:		
Operating	64.52	0.039
Maintenance	25.80	.016
Supervision	9.68	.006
Total labor	100.00	.061
Average tons per man-shift		133.32
Labor, percent of total cost		29.21
(B) Power and supplies:		
Total power, kilowatt-hours per ton		17.36
Dry crushing 1.79		
Wet grinding 8.82		
Jigging .... .40		
Flotation and drying 4.78		
Chat disposal .... -		
Water supply 1.57		
Power and supplies, percent of total cost		70.79

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